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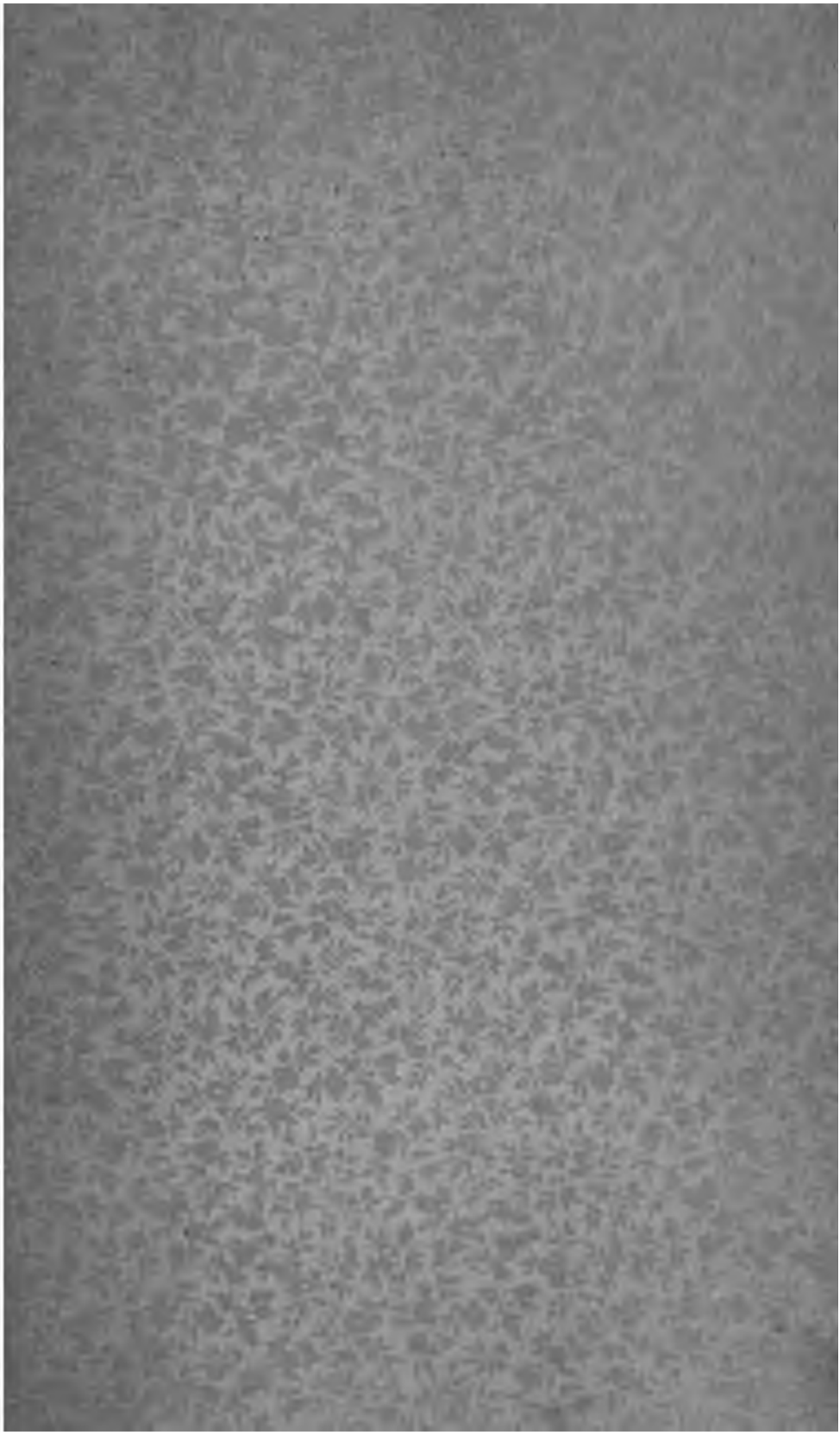

THE

MINE FOREMAN'S HANDBOOK

OF

PRACTICAL AND THEORETICAL INFORMATION

ON THE

OPENING, VENTILATING, AND WORKING OF COLLIERIES.

QUESTIONS AND ANSWERS ON PRACTICAL AND THEORETICAL COAL MINING, DESIGNED TO ASSIST STUDENTS AND OTHERS IN PASSING EXAMINATIONS FOR MINE FOREMANSHIPS.

BY

ROBERT MAUCHLINE,

EX-INSPECTOR OF MINES IN PENNSYLVANIA; LATE INSPECTOR OF MINES FOR THE TENNESSEE COAL AND IRON CO.

THIRD EDITION, THOROUGHLY REVISED AND ENLARGED,

BY

F. ERNEST BRACKETT,

AMERICAN MINING ENGINEER, AND DESIGNER OF COLLIERY PLANTS.

SECOND THOUSAND.

ILLUSTRATED BY ONE HUNDRED AND THIRTY-FOUR ENGRAVINGS.

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PREFACE TO THE THIRD EDITION.

THERE having for some time past been a very urgent demand for a new edition of the late Robert Mauchline's MINE FOREMAN'S HANDBOOK, and the present editor having undertaken the preparation of that edition, he would here state that he has endeavored to bring the book into touch with the more modern processes of Coal Mining; at the same time retaining the author's methods and views which in the past have made the book so popular among that class of workers for whom it was prepared.

With this end steadily in view, new material has been added on Safety Lamps, Coal Dust Explosions, M. Murgue's Values for the Coefficient of Friction of Air Currents in Mines, Size of Ventilating Fans, Boilers, Haulage, the Flow of Water in Pipes, Culm Flushing, etc.

The chapters on Ventilation have been thoroughly revised and corrected. The Miscellaneous matter in the old Chapter VII has been sorted and classified to make reference more convenient. The calculations throughout the book have been checked, and are believed to be substantially correct. Arithmetical work has been performed on the slide rule.

With these additions and revisions, the editor believes that the edition now presented to the public will prove as useful and as popular as the former ones.

The volume has been supplied with a full table of contents, and a good index, thus rendering reference to any subject in the book prompt and satisfactory.

F. ERNEST BRACKETT.

CUMBERLAND, MARYLAND, May 1, 1905.

AUTHOR'S PREFACE TO THE SECOND EDITION.

THERE are few industries in which theoretical and practical knowledge is of more importance than in COAL MINING.

The danger to life and the risk to property are so great that several States have passed laws to protect the one, and capital has introduced the most advanced business methods to counteract the other.

The difficulties encountered in mining are increasing with the depth, and the fastidiousness of the market with the competition, so that the management of a modern colliery requires more than ordinary intelligence.

In the interest of safety the laws of some States demand that those having the care and direction of mines shall hold Certificates of Competency, and Boards of Examiners are appointed to examine candidates and grant certificates to those who pass a satisfactory examination, the aim being to raise the standard of intelligence among those on whom the safety of others largely depends.

Those whose ambition it is to become mine foremen are generally men who have spent their youth in the mines and whose education has been neglected. Therefore the preparation necessary to face a Board of Examiners is to them a work of great magnitude.

The writer, who belongs to the above class, having had considerable practical experience, and acquired some knowledge as to the requirements of the average candidate for a certificate, has, during a period of leisure, prepared the following questions and answers with a view to assist others who may be less informed, and who desire to improve themselves.

The range of subjects in these questions and answers is such as, in the opinion of the writer, to embrace material from which

answers may be framed to nearly all questions likely to be asked by the average Board, and in language calculated to reach the class for which it is intended.

In these questions and answers the educated Mining Engineer may find much for criticism. Our reply to such criticism is, if we, like him, had had the advantage of early training, he would perhaps find less to criticise.

If in the answers given the earnest candidate finds hints which assist him in obtaining a certificate, the aim of the author will be accomplished, and his labor will therefore not have been in vain.

Since the exhaustion of the first edition there have been many inquiries for the book, and it has, therefore, been decided to issue a second edition under the impression that it will fill a want which may increase with the spread of mine legislation in the coal-producing States of the Union. For this purpose the original edition has been thoroughly revised and corrected, and it is hoped that together with the new features which have been introduced the candidate for a certificate will find this new edition a decided improvement on the first.

ROBERT MAUCHLINE.

WOODLAWN, JEFFERSON COUNTY, ALABAMA,
July 10, 1893.

Note by the Publishers. May 6, 1905.—Robert Mauchline was born at Larkhall, Lanarkshire, Scotland, July 27, 1837, and died at Blocton, Bibb Co., Ala., November 23, 1899. At the time of his death, he was Inspector of Mines to the Tennessee Coal and Iron Co., for the Blocton division.

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THE

MINE FOREMAN'S HANDBOOK.

CHAPTER I.

MINE GASES.

Question 1.—What gases destructive to life, or injurious to health, are encountered in coal mines?

Answer.—Light carburetted hydrogen gas, called fire-damp; carbonic acid gas, called black-damp; carbonic oxide gas, called white-damp; sulphuretted hydrogen gas, sometimes called stone-damp; and after-damp, the result of the explosion of a mixture of air and fire-damp.

Ques. 2.—Which, or how many of these gases have you encountered in the mines?

Ans.—All, except the sulphuretted hydrogen, of which I have no personal experience.

Ques. 3.—How would you detect fire-damp in a mine?

Ans.—In collieries of the present day, with the improved methods of ventilation generally at command, the only safe detector is the safety-lamp. In days gone by, when the atmosphere of a mine became so charged with gas as nearly to reach the explosive point, miners used to place their lamps behind their fingers and look at the flame through the small aperture thus formed, much like a person investigating a beam of light through a key-hole, the object being to determine the length of the flame and to see if a halo or cap could be detected, and thus tell when to retreat. But with modern ventilation the mixtures of gas and air are either explosive or are carried off by the air-current, so that "trying the lamp" is now obsolete.

Ques. 4.—What is the danger attending the presence of fire-damp in mines?

Ans.—Fire-damp is one of the greatest dangers encountered in mining. It issues from the pores of the coal, from seams in the surrounding strata, and sometimes it will burst forth in blowers from cavities in the rocks, under pressure like steam from a boiler. These sudden outbursts are extremely dangerous, because they fill the excavations in the vicinity so quickly that persons are liable to be suffocated, and, as the gas spreads and mixes with the air, the result is an atmosphere liable to explode. When pure or nearly so it is not explosive, but as it mixes with the air of a mine it renders the whole surrounding atmosphere dangerous, and when it comes in contact with a light an explosion is the result. Being only half the weight of common air, it fills the upper portions of a mine and gradually diffuses if left undisturbed, so that it is always found in the highest excavations. The principal object in modern mine ventilation is to remove it by air-currents, or to so dilute it as to render it non-explosive. Fire-damp is not a very poisonous gas. We are told that 1 volume of this gas mixed with 2 volumes of air can be breathed for a short time, and that it takes 50 per cent. to produce asphyxiation.

Ques. 5.—Give the proportion by volume of air and gas that renders the mixture explosive; also the percentage of gas that can be detected with a safety-lamp, and what is the most violently explosive mixture?

Ans.—About 30 volumes of air and 1 of gas, up to 15 of air and 1 of gas, will not explode, but shows an elongation of the flame of a lamp with a faint blue halo surrounding it. Twelve or thirteen volumes of air to 1 of gas explodes feebly with a large mass of reddish-colored flame. Nine or ten volumes of air and 1 of gas is probably the most violently explosive mixture. As we add more gas the mixture becomes less explosive, until when there are only 5 volumes of air to 1 of gas it will not explode, but will extinguish lights, and, of course, would soon extinguish life. When we encounter explosive mixtures in the mines, we must judge by the action of the flame in the lamp, etc., and by the general conditions as to the violence of the combustion inside the gauze.

Ques. 6.—Would a mixture of fire-damp and air, just below the explosive point, become explosive when suddenly compressed, as by the discharge of a heavy shot, a sudden and extensive fall, or a heavy explosion in adjacent workings?

Ans.—I believe that experiment has determined this to be a fact beyond dispute, the probable cause being the elevation of temperature by compression, in conjunction with the closer proximity of the molecules favoring the ignition of the oxygen.

Ques. 7.—Suppose a gallery, or chamber, in a mine of uniform section contained an accumulation of explosive mixture which filled it 6 feet back from and across its whole face; if ignited, what distance back would the flame reach, presuming it to extend no further than the distance due to the expansion of the gas when converted into flame?

Ans.—That cannot be determined exactly, as its expansion depends on the percentage of gas present, or the amount of fresh air available to supply oxygen, etc. But with violently explosive mixtures chemists state that the expansion will give about 8 times the volume of flame as of gas. Therefore, in this question, the flame would reach $6 \times 8 = 48$ feet back.

Ques. 8.—How do you detect black-damp (CO_2)?

Ans.—By watching the flame of the lamp. When the air is mixed with black-damp the lamp burns dimly, and will sometimes be nearly extinguished. When a wave of fresh air passes over it, it will brighten up again. When there is considerable black-damp mixed in the air, by waving the lamp it is apt to become extinguished; the amplitude and velocity of the motion which the flame can resist are indicators of the percentage present. This kind of knowledge is the result of long experience.

Ques. 9.—How is black-damp generated in mines?

Ans.—Black-damp is produced by decaying organic matter, by the breathing of men and animals, and, in connection with carbonic oxide (CO), from the combustion of all substances containing carbon. It appears to be held in the strata in large quantities, as it often accumulates in mines where none of the above causes are sufficient to account for the large volumes sometimes present.

Ques. 10.—What percentage of black-damp will vitiate a given volume of air so as to make it dangerous?

Ans.—It is dangerous to breathe air containing 8 per cent. and lights are extinguished in air containing 10 per cent. of this gas, and smaller percentages are injurious in proportion. Being $1\frac{1}{2}$ times the weight of common air, it finds the lowest levels and requires powerful currents to remove it. Persons inadvertently entering a body of this gas are liable to be overcome and suffocated, and air containing small quantities causes drowsiness and soon undermines the constitution. The reason of its injurious effects on the human system is the close affinity between the oxygen and the carbon. The lungs being unable to separate them, there is no oxygen to sustain life.

The presence of $\frac{1}{7}$ of this gas in an atmosphere of fire-damp will render it non-explosive. This is called in mining language "damping the gas," and arguments can and have been advanced to prove that if it were not for the black-damp present in the extensive wastes of some collieries, the whole of the abandoned workings would become a dangerous magazine of explosive mixture, where falls, etc., have made it almost impossible to reach it by ventilation; thus demonstrating that the black-damp was the least of two evils where the accumulations could not be reached by any system of ventilation.

Ques. 11.—What produces white-damp (CO) in dangerous quantities in a mine?

Ans.—I have never been aware of any danger arising from this gas in a mine, except from mine fires, locomotive engines used for hauling, and from boiler fires, but very seldom from ventilating furnaces, although the combustion of all the explosives produces it. I am not aware of any serious accidents occurring through it, except from the combustion of coal or wood inside of mines. Carbonic oxide or white damp is often present in the after-damp of an explosion. It is then due to the incomplete combustion of the fire-damp or other carbonaceous matter caused by the lack of sufficient air.

This is perhaps the most dangerous of all mine gases, especially to those who have had no experience with it.

Very little experience is required to detect fire-damp with a safety-lamp, and black-damp very quickly makes its presence known by quenching the light. But a person might enter an

atmosphere vitiated by white-damp, and fall insensible before the lamp gave any indication of danger.

Three per cent. mixed in the atmosphere of a mine will cause death, so that whenever the smell is detected it is time to retreat to fresh air. Some writers consider ½ per cent. of this gas sufficient to cause death.

One volume of this gas to seven of air when ignited will become a mass of flame, but the explosive force is not great, except when large quantities are ignited.

Ques. 12.—How do you detect white damp (CO)?

Ans.—By its peculiar odor, which can be detected at once by those who have had to deal with it, and who know the danger of remaining in it after detecting its presence.

Ques. 13.—How is sulphuretted hydrogen (or stone-damp) produced, and what is the danger attending its presence in mines?

Ans.—Sulphuretted hydrogen gas (H_2S) is produced where sulphur pyrites are decomposing in contact with hydrogen. This gas is sometimes found in abandoned unventilated workings containing stagnant water. When mixed with air it burns with a blue flame. This gas can be so mixed with air that lights will burn while life will become extinct; it can thus be only detected by its odor, resembling that of rotten eggs. This gas is seldom met with in coal mines, and, therefore, is not included in the common dangers. I have never seen or encountered it in mines during a varied experience of many years.

Ques. 14.—What is after-damp, and how is it produced?

Ans.—After-damp is the resultant mixture left after an explosion of fire-damp, and contains about 7 parts of free nitrogen (N), 1 part of carbonic acid gas (CO_2), and 2 parts of steam, in 10 volumes. The steam generated by the chemical union of the hydrogen contained in the fire-damp, with the oxygen of the air, and by moisture being subjected to the intense heat of the explosion, soon condenses, leaving the mixture about 7½ parts of free nitrogen (N) and 1 part carbonic acid gas (CO_2) in 8½ parts. Or, practically, the after-damp *is the air with the oxygen burned out by the explosion*, mixed with a little black-damp. The quality of the mixture will, of course, vary with the percentage of the mixture which exploded.

The above figures are those found by experimental chemistry; but as the gases found in mines are generally mixed, these calculations are not strictly correct for gases as we encounter them in mining operations.

Ques. 15.—What are the principal causes of vitiation of the atmosphere of a mine?

Ans.—Fire-damp (CH_4) given off from the coal, blowers of the same gas from sudden outbursts where it is held under great tension, and feeders from seams in the floor and roof. Black-damp (CO_2) given off from the coal principally in old and pillar workings, smoke from the use of explosives and from the burning of lamps. The breathing of persons and animals, and, where fire is present, the production of carbonic oxide (CO) as the result of combustion.

Ques. 16.—How do you ascertain the weight per cubic foot of these gases, or their specific gravity as compared with air?

Ans.—In chemistry the atomic theory starts out by assuming that simple gases are constituted of atoms which are the limit of division; that these atoms are equal in number and magnitude in any given volume, and that the difference in weight of different gases is due to the difference in weight of the atoms, and hydrogen, being the lightest, is taken as a unit. The atomic weight of a gas is, therefore, its weight as compared with an equal volume of hydrogen, and is given in works on chemistry as follows:—

Hydrogen,	symbol	H	atomic	weight	1
Carbon,	"	C	"	"	12
Nitrogen,	"	N	"	"	14
Oxygen,	"	O	"	"	16
Sulphur,	"	S	"	"	32

These letters stand for the name of the elementary gas or substance, and the figures for their relative weights. Carbon and sulphur are not gases, but are found in chemical combination with the gases found in mines.

If we now carefully commit to memory that the atomic weights of hydrogen (H), carbon (C), nitrogen (N), oxygen (O), and sulphur (S) are 1, 12, 14, 16, and 32, respectively, and also that by experiment a cubic foot of hydrogen weighs

0.0056 of a pound, from these we can tell the weight per cubic foot of any mixture or combination of these gases.

We must also remember that where these gases are mixed mechanically, as in common air, we divide the sum of the atomic weights by the number of volumes; but where the gas is a chemical combination we divide the sum of the atomic weights by 2, because when two of these elementary substances combine chemically to form the gases found in mines, in the act of combining they shrink into the same number of volumes that hydrogen would occupy under the same temperature and pressure. To make this more easily understood, let us find the weight of these gases. Now, in order to compare them with common air, we must find its atomic weight.

Atmospheric air is a mechanical mixture of 1 part oxygen and 4 parts nitrogen, with a trace of carbonic acid gas, which may be omitted in our question, the trace being so slight. The formula is:—

$$O + 4\,N = \frac{16 + (4 \times 14) \times 0.0056}{5} = 0.08064$$

as the weight of a cubic foot of air. This means that the atom of oxygen (O), having a weight of 16 when hydrogen is equal to 1, has been mixed with 4 atoms of nitrogen (N), having a weight of 4 times 14, and the sum of their atomic weights is $14 \times 4 + 16 = 72$, but as there are 5 atoms in every volume of the mixture, we must divide the sum of atomic weights by 5 to obtain the average weight as compared with the unit, hydrogen—

$$\frac{72}{5} = 14.4$$

as the atomic weight of air, being an average of the atomic weights of nitrogen and oxygen.

A cubic foot of hydrogen under ordinary conditions has been found to weigh 0.0056 of a pound, and its weight is to air nearly as 1 is to 14.4.

Therefore, 1 : 14.4 :: 0.0056 : 0.08064 lb. per cubic foot, or multiply the atomic weight by the weight of a cubic foot of hydrogen. From the above we can now find the atomic weight, and from it the weight per cubic foot of the gases found in mines, as required by the terms of this problem.

Fire-damp, or light carburetted hydrogen, is CH_4 combined, which means that 1 atom of carbon is equal to 12 where hydrogen is 1, and 4 atoms of hydrogen equal 4 times 1, having combined, which would be equal to $12 + 4 = 16$, the sum of the atomic weights of these atoms. But as these 5 atoms or volumes have shrunk in the chemical action of combining into the space occupied by 2 volumes or atoms of hydrogen, if we wish to compare them with the unit, hydrogen, we must divide them by 2 to find the atomic weight of the combination, and multiply it by the weight of a cubic foot of hydrogen to find its weight per foot.

$$CH_4 = \frac{12 + 4}{2} = 8, \text{ the atomic weight.}$$

$$8 \times 0.0056 = 0.0448 \text{ lb. per cubic foot.}$$

We have found its weight per foot; let us now find its weight as compared with air, or its specific gravity. We find that the atomic weight of air is 14.4; therefore, if we wish to compare this gas with air, we have the proportion

$$14.4 : 8 :: 1 : 0.555;$$

or divide the atomic weight by the atomic weight of air, and the formula becomes, for fire-damp:—

$$CH_4 = \frac{12 + 4}{2} \times 0.0056 = 0.0448 \text{ per foot.}$$

$$8 \div 14.4 = 0.555, \text{ the specific gravity.}$$

This law of shrinking into 2 volumes, or its equivalent of hydrogen, causes us to use the weight of a foot of hydrogen as the unit; but hydrogen would not be a convenient comparison for specific gravity, especially in mines, because we wish to become familiar with the weights of gases as compared with air, so as to be able to judge of the relative positions they will assume—whether on the bottom or next the top when not disturbed by currents or diffusion. The same process finds the same result for the other gases.

$$CO_2 = \frac{12 + (2 \times 16)}{2} \times 0.0056 = 0.1232 \text{ lb.}$$

the weight per cubic foot; and

$$\frac{22.0}{14.4} = 1.527, \text{ the specific gravity.}$$

White-damp, or carbonic oxide (CO) combined, is

$$CO = \frac{12 + 16}{2} \times 0.0056 = 0.0784 \text{ lb.}$$

per cubic foot, and

$$\frac{14.0}{14.4} = 0.972, \text{ the specific gravity.}$$

Stone-damp, or sulphuretted hydrogen (SH_2) combined, is

$SH_2 = \frac{32 + 2}{2} \times 0.0056 = 0.0952$ lb. per cubic foot; and

$$\frac{17.0}{14.4} = 1.1805, \text{ the specific gravity.}$$

The accompanying table (A) will be found useful, as it brings the gases with their combinations, as found in mines, before the eye, for comparison. By studying the comparative figures we will soon commit to memory the formula necessary to find the weight of any of these gases.

TABLE A.

Common Name.	Chemical Name.	Symbol.	Formula.	Atomic Weight.	Specific Gravity. Air = 1.	Weight per cubic foot in lbs.
	Hydrogen	H		= 1		0.0056
	Carbon	C		= 12		0.0 72
	Nitrogen	N		= 14		0.0784
	Oxygen	O		= 16		0.0896
	Sulphur	S		= 32		0.1792
Atmosphere	Air	O + 4N	$\frac{16+(4\times14)}{5}=$	14.4	1.0	0.08064
Fire-damp	Light carburetted hydrogen	CH_4	$\frac{12+(4\times1)}{2}=$	8	.55	0.0448
Black-damp	Carbonic acid gas	CO_2	$\frac{12+(2\times16)}{2}=$	22	1.527	0.1232
White-damp	Carbonic oxide	CO	$\frac{12+16}{2}=$	14	.972	0.0784
Stone-damp	Sulphuretted hydrogen	SH_2	$\frac{32+2}{2}=$	17	1.1805	0.0952
After-damp	Mixture	$CO_2+7\frac{1}{2}N$	$\frac{22+(7.5\times14)}{8.5}=$	14.94	1.0375	0.0806

To find specific gravity, ÷ 14.4.
To find weight per cubic foot, × .0056.

The following is the result of experiments made by M. Galloway, with the safety-lamp, with different mixtures of air and gas:—

The wick of the lamp having been carefully trimmed, was drawn down until the flame presented the appearance of a small blue hemisphere about $\frac{1}{8}$ of an inch high and $\frac{1}{4}$ inch in diameter at the base, having a conical speck of yellow in the middle near the top.

A mixture of 1 part gas to 16 parts air gave a wavy spindle-shaped blue cap $3\frac{3}{8}$ inches high.

1 gas to 18 air—similar cap 2 inches and more steady.

1 gas to 20 air—cap $1\frac{1}{16}$ inches high, with parallel sides $\frac{3}{8}$ of its height and drawn to a point at top; this cap was steady and more distinct than the others.

1 gas to 25 air gave a conical cap $\frac{1}{2}$ to $\frac{5}{8}$ inch high.

1 gas to 30 air gave a conical cap $\frac{3}{8}$ inch high.

1 gas to 40 air gave a conical cap $\frac{3}{16}$ to $\frac{1}{4}$ inch high.

1 gas to 50 air an exceedingly faint cap $\frac{1}{8}$ inch high, the top having the appearance of having been broken off.

1 gas to 60 air. Hardly possible to distinguish anything above the small oil flame. These heights were determined from experiments made on a mixture of pure air and pure marsh gas. In practice a cap $1\frac{3}{16}$ inches and less has been found to indicate explosive conditions especially in the presence of coal dust. The height of these caps for the same percentage of gas varies with the kind of lamp and also with the oil or other fluid burned.

Ques. 17.—What is meant by the term diffusion of gases?

Ans.—Diffusion is a natural tendency in all gases, which do not combine chemically, to mix intimately. Experiment has shown that a light gas, such as hydrogen, will diffuse much quicker than a heavy gas, like carbonic acid or oxygen, and that the ratio of diffusion is inversely as the square root of the density. If two mine chambers of an equal capacity were filled, one with carbonic acid gas and one with light carburetted hydrogen, then the light gas would diffuse and become diluted with air in much less time than the heavy gas, all other things being equal. The density of fire-damp being 8 and the density of black-damp being 22, then $\sqrt{8} = 2.82$ and $\sqrt{22} = 4.69$, so that the time of their diffusion would be about as $2\frac{3}{4}$ is to $4\frac{1}{2}$, or nearly 2 to 1.

Therefore air will diffuse with, or dilute an accumulation of, fire-damp much quicker than an equal accumulation of black-damp. This law of diffusion has considerable effect in the formation of explosive mixtures in mines, or parts of mines, not reached by the ventilating current. Suppose that along the level gangway, or entry, of a mine, a number of chambers or working places are driven up grade, parallel with the dip, with openings in the pillars which separate them, through which a very feeble current of air is passing; let each chamber be, say, 20 yards in advance of the headings through which the air is moving, and suppose the coal face of each chamber to be giving off a little gas: the diffusion going on will mix the gas down to where the current carries it off through the pillars. The mixture at the face where the gas is issuing will contain the greatest percentage of gas, which will gradually decrease down to the point where the pure air is passing. If the quantity of gas given off is just sufficient to balance the diffusion, the percentage will remain constant; if the effusion ceases, the gas will be carried off; if the effusion slightly increases, an explosive mixture will form at the face. Now let us suppose the air at the faces of these chambers to contain about 1 per cent. of gas, or just as much as can be mixed in the air without being detected with a safety-lamp, and mixed with 1 per cent. where it is carried off. When this air containing 1 per cent. reaches the second chamber, the diffusion will be between air containing 1 per cent. from each. This may raise the percentage to nearly 2. The gas from the third chamber will have to diffuse with air containing 2 per cent., and so on; so that by the time the fourth or fifth chamber is reached the mixture may be explosive, therefore the slightest change in the quantity of gas given off may result in forming dangerous accumulations. Where fire-damp has to be removed downwards with a feeble current just sufficient to balance the upward tendency of the gas, diffusion will mix the air and gas, and, if the quantity reaches about 7 per cent., the whole will become explosive, even if the chamber was driven down grade. This has occurred in rare cases, where an airway having a very feeble downward current was connected with places giving off large volumes of gas.

This law of diffusion is therefore a power in the formation of dangerous accumulations of gas, in mines where the ventilation is small in volume and imperfect in distribution, and is a factor worthy of attention in the ventilation of mines.

Ques. 18.—What effect has coal dust in mines?

Ans.—Besides fouling the air it often plays a very important part in explosions. It has been found that a 2 per cent. mixture of fire-damp and air becomes explosive when finely divided dust is present in suspension. It has also been found that coal dust in suspension may be explosive even if no fire-damp is present; but in that case it cannot be ignited by a naked light or ordinary flame. Badly tamped or misplaced shots or local explosions of fire-damp frequently ignite dust explosions of this class; and an explosion once originated may feed upon the dust raised or encountered by it, and thus travel long distances. I believe all coal dusts are explosive, but some are much more so than others. In this country some soft coals give much trouble in this way. In the Anthracite region, under existing conditions, dust explosions are practically unknown.

Ques. 19.—How do you account for the explosion of dust?

Ans.—Opinions differ somewhat as to the exact nature of a dust explosion. The heat of the initial explosion, acting upon fine particles of dust, probably drives off carburetted hydrogen, which is present as a part of most coals; and, acting directly upon the fixed carbon of the coal, forms carbon monoxide by uniting it with the oxygen of the air. Either of these gases explodes on uniting with a fresh supply of air in the presence of flame, only to repeat the process with a new supply of dust. Whatever is the exact nature of the explosion, we know there are carbon and hydrogen present in the dust and oxygen present in the air, which is all that is necessary for combustion, and experiment and experience have shown that under favorable conditions, when the dust is finely divided and properly ignited, an explosion occurs. With reference to blasting it has been pointed out that much dust is produced by the shot itself in a confined space suitable for the production of carbon monoxide. Also the powder itself nearly always produces this gas. These two influences are undoubtedly great factors in igniting the surrounding dust.

Ques. 20.—How would you provide against dust explosions?

Ans.—The dusty roads of the mine should be regularly sprayed with water conducted by pipes and hose provided for that purpose. The working faces should be kept as free as possible from dust by loading all slack with the coal. I would make careful tests with a sensitive lamp to ascertain the presence of fire-damp. If fire-damp is present, it should be kept below 1 per cent. by diluting it with a strong ventilating current; and safety lamps should be used. Shots should be carefully placed and tamped so that no "blow out shots" occur, as these prolong the flame of the charge. They should be fired by a special set of men after the miners have left the mine. No second shot should be fired in an unsuccessful hole as the coal has probably been much shattered inside by the first shot, and the second one may therefore produce a large amount of dust. If the surrounding space is very dusty it should be sprayed before firing, care being taken to spray sides and timber as well as floor if necessary. I would use some explosive which experiment has proven to be less likely to ignite the surrounding atmosphere than powder.

CHAPTER II.

SAFETY-LAMPS AND INSTRUMENTS.

Ques. 1.—What is the principle upon which the safety-lamp is constructed?

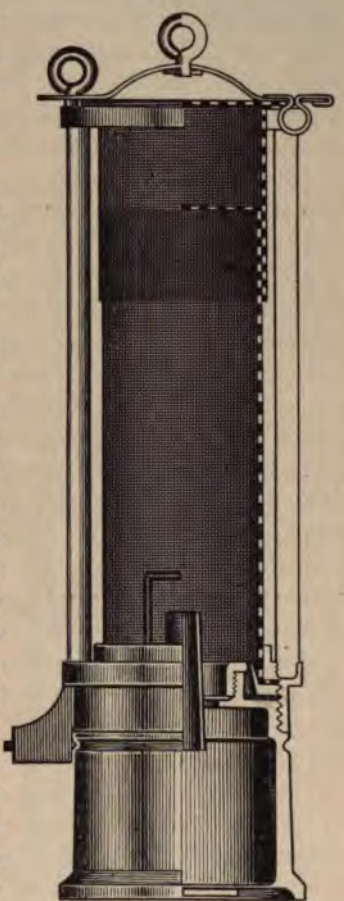
Ans.—The claim to safety in all the so-called safety-lamps which I have seen is involved in the principle of preventing the flame from coming in contact with the surrounding gas, by enclosing it in a cylinder of fine wire gauze. It has been found by experiment that flame will not pass through small apertures except by pressure. The theory is that flame in coming in contact with the surface of the metal and being finely divided, as with fine wire gauze, is cooled below the temperature necessary to ignite gas; therefore when the gas fires inside the gauze cylinder and fills it with flame, the chill or cooling influence of the wire absorbs as much of the heat as brings the flame below the temperature necessary for ignition. The wires being about 28 to the inch, make the apertures so small that each little column of flame is affected to its center and extinguished, unless forced through by pressure. Experiment has shown that a current of explosive mixture with a velocity of about 8 feet per second will drive flame through the wire gauze, so that the lamps are only safe in currents of less than 8 feet per second. Even when the gauze becomes red hot by the combustion of the gas inside the lamp, it is still cooler than the flame, and gas will not ignite except by direct contact with flame. Such an experiment, however, would be extremely dangerous, as the gauze would soon be destroyed and an explosion would result. Therefore, whenever the gauze of a safety-lamp becomes hot the lamp must be removed to a pure atmosphere and allowed to cool. When the gauze has been repeatedly heated to a high temperature it should be removed and the lamp fitted with a new one.

Ques. 2.—With how many different kinds of safety-lamps are

you familiar, and what is the principle upon which they are constructed?

Ans.—There are a great many different kinds of safety-lamps in use, many of them differing only in detail. Also there are many different forms of some of the modern lamps. I am familiar with the "Davy," "Stephenson," "Clanny," "Mueseler," "Marsaut," "Evan Thomas," "Ashworth-Hepplewhite-Gray," "Wolf," "Hughes Bonneted Fireboss" and "Pieler" lamps; and will describe some of the typical forms of them.

Fig. 1.



The Davy lamp (Fig. 1) having simply the flame inside the gauze cylinder is a good detector of gas. Being surrounded by gauze only, the gas reaches the flame direct and can be easily seen; it is thus very sensitive to gas, but the exposure of the flame makes it easily extinguished in strong currents of air. When the lamp is carried against the current in an explosive mixture, if the speed of the current and the opposite speed of the lamp, added together, amount to 7 or 8 feet per second, the flame of the gas inside may be forced through the gauze and fire the mixture, or the rush caused by a heavy fall may have the same effect. In experienced hands it is a good lamp. It is light, simple and durable, but not fit to be placed in inexperienced hands, being the most sensitive and also the most dangerous.

The "Stephenson" lamp is fitted with a glass cylinder inside the wire gauze; the upper end of the glass is covered with a perforated copper cap, the bottom is made tight against a metal ring, so that air from the outside cannot enter through the gauze and down to the flame. The air to support combustion enters through small holes drilled in the brass ring on which the glass rests, and also through some meshes of the gauze which projects down below the level of the metal ring. When the lamp is burning it is thus fed with air at the bottom, and the products of combustion and heated air cause an upward current through the glass cylinder, so that gas cannot enter in a downward direction into the lamp against the current. The holes below admit just sufficient air to maintain the flame. When gas enters and fills the gauze with flame, the sudden increased demand for air cannot be supplied, and the lamp is extinguished for lack of oxygen. It is, therefore, a self-extinguishing lamp, and cannot burn in an explosive mixture. Its faults are that the light having to pass through the glass in addition to the gauze is obstructed, and it gives a very dim light, it has to be much larger and somewhat heavier than the Davy, is more complicated and expensive, and the glass cylinder is easily broken. When the lamp is tilted, even in a pure atmosphere, it is apt to go out, and when it is placed in gas the same effect is produced. This is its strong claim to safety, but does not suit in examining for, or as a detector of, gas, as it is extremely dangerous to lose light in some situations when examining for gas. When used in very dry places the dust collects on the glass, and, as it is inside the gauze, it cannot be wiped off except by opening the lamp. It is now out of date.

The "Clanny" lamp, Fig. 2, is an improvement on the Davy so far as light is concerned, the flame being surrounded with a strong cylinder of glass, above which is fixed the wire gauze cylinder. The Clanny gives a much better light than the Davy or Stephenson, and the flame being down inside of the glass cylinder is not easily blown out by strong currents. When placed in an explosive mixture where there is little or no current, it is generally extinguished from the carbonic acid gas produced by the explosive gas burning in the lamp filling the

glass cylinder and smothering the flame. It is well adapted for level workings. It gives a good light, but is more apt to go out if filled with gas than the Davy, therefore better adapted for

FIG. 2.

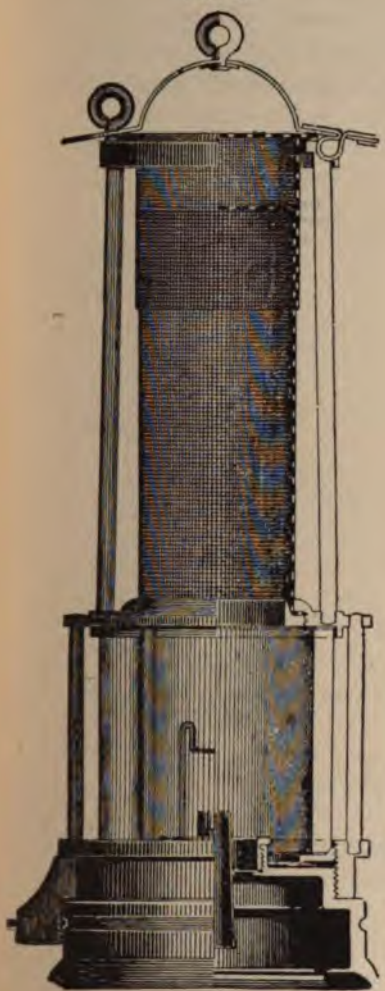
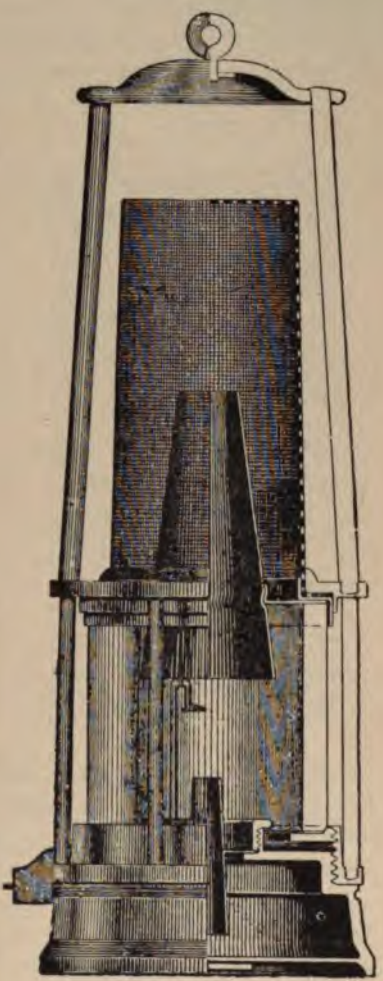


FIG. 3.



miners or drivers; but on steep angles it is too heavy and the glass is more liable to break from striking against the manway.

There is also a prejudice against the lamp because of the liability of the glass cylinder to crack from drops of water falling on it when heated, or from sudden contraction by cold currents of air. I have never heard of accidents from this cause, and even a cracked glass will not cause an explosion, because the top and bottom of the cylinder are fixed in a ring to keep it firmly in place until the lamp is carried out and a new cylinder put in.

The "Clanny" is very little safer than the "Davy" where

FIG. 4.

FIG. 5.

exposed to currents of air. It is not considered safe at over 10 feet per second velocity of current. It is sometimes supplied with a cylindrical cover or "bonnet" for the gauze, which very much increases its safety in air currents. It is then known as a "Bonneted Clanny." In the "Clanny" lamp the air for combustion has to enter over the top of the glass and go down inside to feed the flame. This sets up a conflicting set of currents in the lamp which injures its lighting powers and makes it liable to smoke.

The "Mueseler" lamp, Fig. 3, is much like the Clanny, only a disk of wire gauze is placed above the flame with a conical tube up through it, the object being to cause the cool air to pass down the outside of the tube and the heated products of combustion to pass up the inside. The Mueseler acts much like the Stephenson when placed in gas. It is a self-extinguisher and gives as good a light as the Clanny, and goes out a little sooner and more quietly. It is undoubtedly a good lamp but is more expensive, a little complicated, and has the same objection as the Clanny, viz.: the cracking of the glass cylinder from drops of water or cold air currents. The Mueseler is considered safe at 20 feet per second velocity of air current. Like the "Clanny" it is sometimes supplied with a "bonnet" to protect the gauze, in which case it is considered safe at about 40 feet per second velocity of air current.

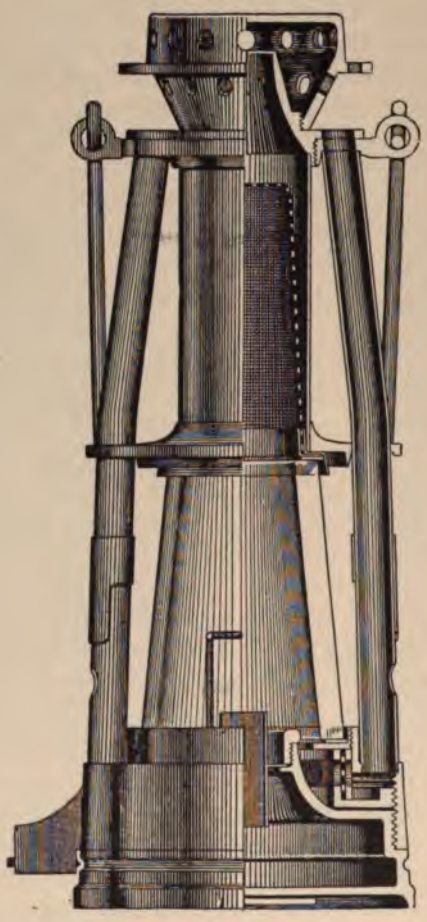
The "Marsaut," Fig. 4, is simply a bonneted "Clanny" lamp with a double or even a triple gauze. Like the "Bonneted Mueseler" lamp, it is safe at about 40 feet velocity of air current.

The "Evan Thomas" lamp, Fig. 5, is much like the "Bonneted Clanny" and "Marsaut." The chief difference consists in a second and inner glass cylinder. The air, entering over the outer glass through the lower part of the gauze, descends between the two glasses and then up under the lower glass to the flame. This arrangement gives a better supply of air to the flame and also keeps the glasses cool. The "Bonneted Clanny" and its modifications; the "Bonneted Mueseler," "Marsaut," and "Evan Thomas" are all good lamps for general use and differ only in detail. The method of feeding the flame with air, however, makes them poor gas-detectors, because the outside air does not become sufficiently mixed with the flame or because they are extinguished too soon.

In order to obtain a better light and also a lamp that could be used for testing for gas with greater security than the "Davy," the "Ashworth-Hepplewhite-Gray" lamp, Fig. 6, was invented. This lamp, like the "Bonneted Clanny," has a glass, surrounding the flame, surmounted by a gauze and bonnet. The air to feed the flame, however, enters through slits in

the standards near the bottom, at the sides of the lamp, passes through a small gauze and enters the lamp below the flame, thus entirely doing away with conflicting currents within and

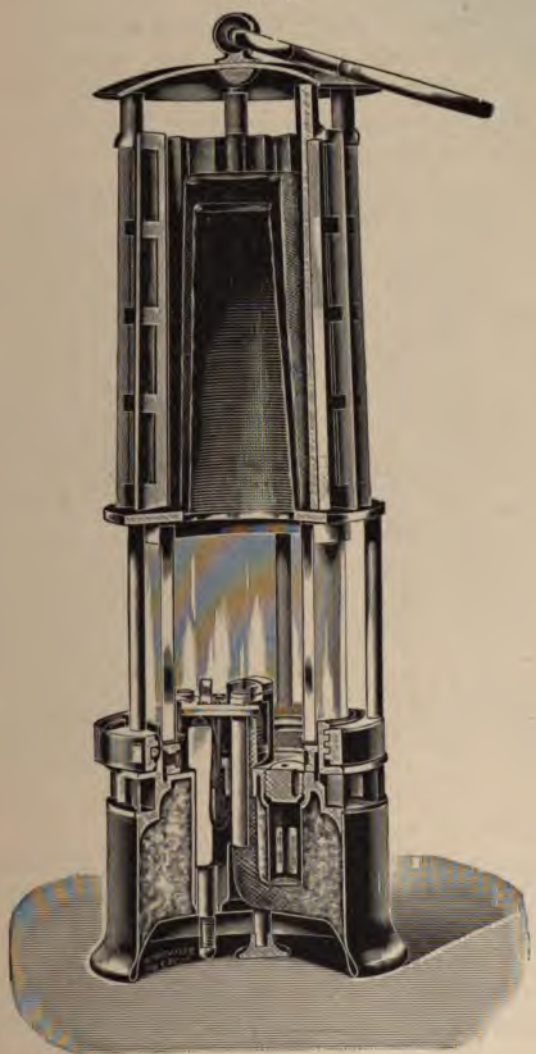
FIG. 6.



giving better conditions for light and the production of the flame cap than could be had in the "Clanny." In order to test for thin layers of fire-damp near the roof without tilting the lamp the standards are made hollow and the lower openings for

taking air are provided with covers which can be closed when desired. The lamp then takes its air from the top instead of

FIG 7.



the bottom. The standards can be prolonged by the addition

of a short tube which is carried by the fire-boss who can thus test for gas lying in extremely thin sheets close to the roof. The glass in the lamp is conical instead of being cylindrical, which enables the light to be thrown upwards, thus illuminating the roof and permitting the miners to make a better examination of it. For testing for small percentages of fire-damp this lamp can be fitted to burn alcohol or hydrogen which, giving a hotter and less luminous flame than oil, gives a greater and more distinct gas cap. It can also be made a self-extinguishing lamp when exposed to explosive mixtures. It is thus safer and gives a better light than any of those previously described.

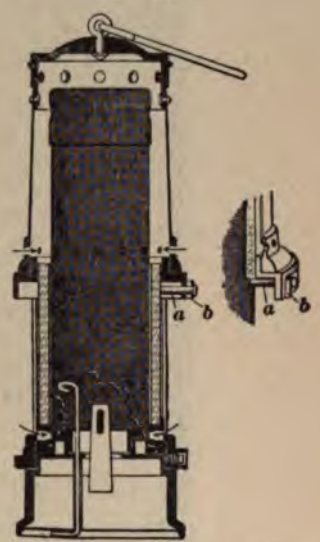
The "Wolf" lamp, Fig. 7, consists of a lower glass cylinder surmounted by a double gauze and bonnet of peculiar design, well adapted to baffle outside air currents. Like the "Ashworth-Hepplewhite-Gray" it takes its air from the bottom. The air first passes through a baffle and then through a lower gauze. The size of all the gauzes is from 33 to 34 meshes per inch. This lamp burns naptha which gives a better light and does not char the wick or dirty the lamp as oil does. Its light is equal to about one sperm candle. The increased heat of the flame makes it an excellent detector of fire-damp. The naptha is absorbed in cotton in the lower part of the lamp. Its use, instead of oil, is said to be cheaper. A percussive arrangement enables the lamp to be relighted without opening. This is a great advantage, as after explosions of fire-damp most lamps are extinguished and the men left to grope their way in darkness to safety. The lamp is locked with a magnetic lock which it is impossible to pick or open in the mine. In the lamp house it is quickly opened by means of a large magnet. When the flame is lowered for a gas test $\frac{3}{4}$ per cent. fire-damp gives a cap $\frac{3}{16}$ inch high above the burner. One and one-half per cent. ($1\frac{1}{2}$) gives a cap $\frac{1}{4}$ inch high; 2 per cent. gives a cap $\frac{1}{2}$ inch high; 3 per cent. gives a cap 1 inch high; 4 per cent. gives a cap $2\frac{3}{8}$ inches high. At 5 per cent. mixture the cap fills the lamp. The makers of the Wolf lamp claim that it will easily withstand a current of 59 feet per second of a 9 per cent. gas mixture. From numerous tests the percussive igniter and the

use of naptha in the lamp have been pronounced safe. Like the "Ashworth-Hepplewhite-Gray," it is a self-extinguisher. It seems to be one of the safest and best lights on the market today, and is gaining in favor.

The "Hughes Bonneted Fire-Boss" lamp, Figs. 8 and 8a, has a full gauze like the "Davy," but is supplied with a bonnet and glass like the "Bonneted Clanny." The glass cylinder can

Fig. 8.

Fig. 8a.

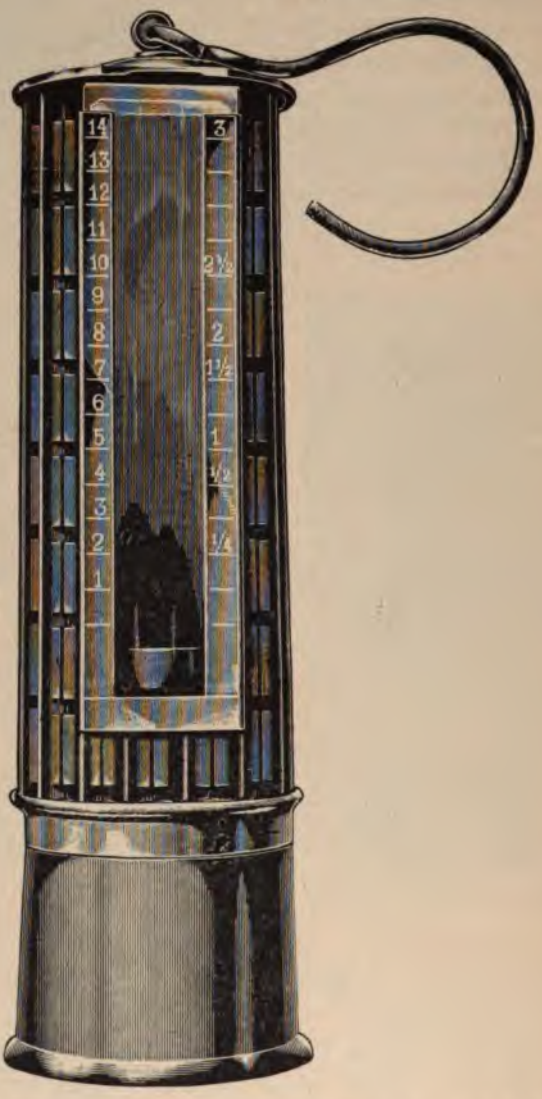


Hughes's Bonneted Fire-Boss Lamp. (Sectional View.)

be raised while testing for gas, thus giving the outside atmosphere free access to the flame. As its name implies, it is designed for gas detection.

The "Pieler" lamp, Fig. 9, is a gas-testing lamp which differs from the "Davy" in burning alcohol and having a scale for reading the height of the gas cap. In its safer forms the gauze is covered with a bonnet, as shown in the figure, where a "Wolf" corrugated shield has been applied. It is an ex-

FIG. 9.

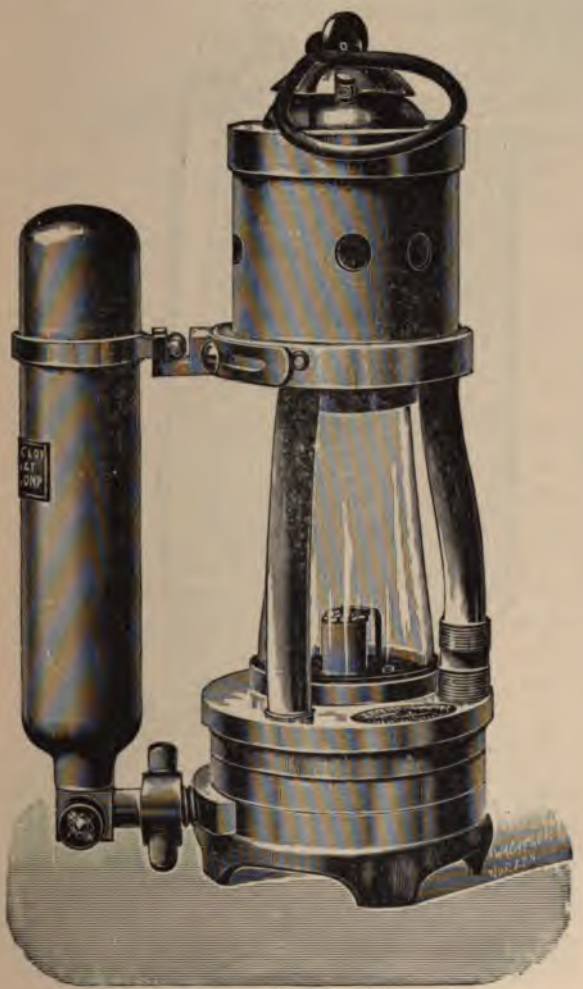


tremely sensitive lamp, giving a cap $1\frac{1}{4}$ inches high for $\frac{1}{4}$ per cent. gas, and $3\frac{1}{2}$ inches high for 1 per cent. gas. At $1\frac{3}{4}$ per

cent. gas the cap reaches the top of the lamp. It is not intended for general work.

Besides these I may mention the "Clowes" gas-testing

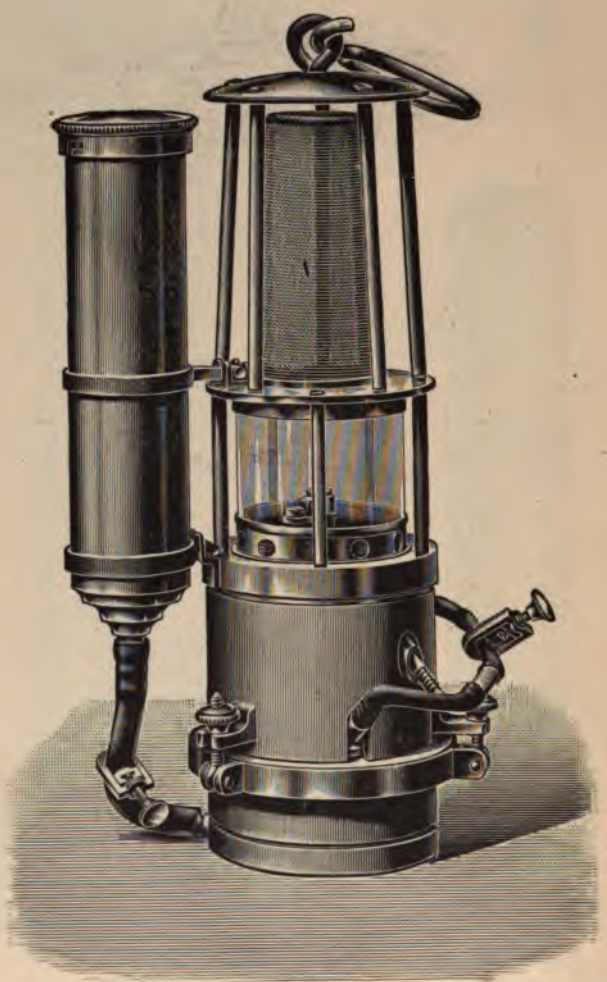
FIG. 10.



lamp, Fig. 10, and "Hempel's" gas-testing lamp, Fig. 11. In the "Clowes" lamp the hydrogen for burning is stored in a

reservoir as shown, and it must be filled before entering the mine. In "Hempel's" lamp the hydrogen is generated by the use of proper chemicals in the lamp itself.

FIG 11.



Ques. 3.—Do you know of any other method of illuminating fiery mines?

Ans.—Incandescent electric lamps have been tried, but I do not approve of their use as a general light, because, being entirely isolated, the nature of the surrounding atmosphere cannot be determined by them. I think one of the most valuable features of a lamp is that it gives warning of the presence of gas. However, in the hands of experienced men, these lamps have proven very valuable in exploring workings after explosions when the after-damp would extinguish other lights.

Acetylene has also been tried as an illuminant, and with some success. At present it has not gained any great use in this country.

Ques. 4.—Why do safety-lamps fail, and under what conditions may they be expected to pass the flame?

Ans.—A lamp faultily constructed, poorly put together, or defective in any way, may pass the flame whenever placed in a fiery atmosphere. In the rest of my answer I will confine myself to lamps which are perfect.

An explosive mixture of fire-damp and air blowing horizontally against the lamp may drive the enlarged flame through the gauze. A downward current may fill the top of the lamp full of fire-damp before ignition takes place, and the violence of the resulting explosion may cause the lamp to pass the flame. An upward current may also cause a lamp to fail. A gauze dirtied by soot or otherwise may begin to glow, and thus ignite the outside atmosphere. It has also been found that the presence of dust very much increases the liability of a lamp to pass the flame. The exposure of a lamp to a blower of pure marsh gas may cause an otherwise safe lamp to fail. Of course a long exposure of any lamp, that is not extinguished, to a fiery mixture, may cause it to become overheated and thus fire.

Ques. 5.—What general principle would guide you in the construction of a safety-lamp?

Ans.—One principle to be aimed at is to have the lamp as small and light as practicable; the oil stock should contain oil sufficient for about 10 hours; the gauze should be clean and the joint clenched tight, so that heating the gauze will not tend to open it. The smaller the space inside the lamp the better, as when the gas ignites there is less flame to heat the wires, and a

small cylinder is stronger. In lamps of the Clanny type the close cylinder surrounding the flame should be short. If it is long, when gas explodes inside the lamp, it strikes upwards or "guns" and may pass through the lamp top. Some minor inventions have been applied to safety-lamps for the purpose of extinguishing the flame, either by the application of snuffers which close over it, or tubes which slide up and cover it, actuated by wires like the picker. These so-called "extinguishers" are a very convenient attachment to extinguish the flame of the lamp. If the miner is overtaken by a body of gas, he can put

FIG. 12.

Anemometer.*

out the light much quicker and safer than by pulling down the wick or smothering it in his clothing. But although it extinguishes the flame of the wick, it does not always extinguish the flame of the gas within the gauze and is therefore unreliable, having the serious fault of creating a false sense of security. Another attachment for the same purpose consists of a tube which slides down over the gauze, completely shutting off the air, which is a much safer extinguisher. All these attachments,

* The cuts, Figs. 12 and 13, are kindly loaned by Messrs. Queen & Co., 1010 Chestnut St , Philadelphia, who are dealers in anemometers and barometers.—H. C. B. & Co.

however, depend on the action of the person using them, and the most reliable "extinguisher" is a lamp so constructed as to be "self-extinguishing."

Ques. 6.—Name and describe the instruments used in examining the condition of the atmosphere of a mine, showing the principle and application of each.

Ans.—The instruments used in connection with the ventilation of mines are as follows: 1st. Safety-lamps of different types used for detecting the presence of gas, which have been briefly described in answer to question number 2; 2d, the anemometer; 3d, the barometer; 4th, the thermometer; 5th, the hygrometer; 6th, the water-gauge.

The Anemometer, Fig. 12, is an instrument for measuring the velocity of an air current. Anemometers are of two kinds. One type consists of a very lightly-made wheel with oblique blades like a screw propeller, connected by clock-work to a dial on which a pointer revolves. The instrument is made so that one revolution round the dial corresponds to one hundred feet of lineal motion of the air current. Therefore, if by a watch it is found to revolve round the dial 5 times a minute, then the velocity is 500 feet per minute, and as the circle is divided into 100 equal parts, the units and tens can be found. Some anemometers have small dials inside the large circle which show the 100s, 1000s, 10,000s, 100,000s and 1,000,000s, so that if the instrument is to be placed in the current for a considerable time, say for 5 or 6 hours, to indicate the average velocity for a given time, then the readings of all the dials are noted, and the instrument started. At the end of the stated time it is taken out and the first reading subtracted from the second, and the difference is the distance, or rather the length, of current that has passed through the instrument, which can be reduced to feet per minute for the whole time; provided the instrument does not remain long enough for more than a million feet to pass. In that case the record would be lost. These instruments are fitted with a disconnector, like a stop-watch. The speed required to start the wheel is added as a constant to the indicated velocity. The other type of anemometer is simply a vane 3 or 4 inches square, pivoted in a frame. When held vertically and at right

angles in a current, the velocity of the air is indicated by the angle to which the vane is raised. On the side of the frame a quadrant scale is fixed on which are marked figures corresponding to the velocity of a current which is strong enough to raise the vane to the point indicated. Thus, if the current can raise the vane up to 700, then it is a current of 700 feet velocity. There is no timing needed, but the unsteadiness of the current causes the vane to oscillate, and the true reading is between the

Fig. 13.

Barometer.

points of oscillation. A level is attached to the frame so that it can be placed vertical. It should also be placed at right angles to the current. The friction of the pivot or hinge is reduced by fine centres and the weight of the vane balanced by a small brass lever and knob on the upper side of the centre, thus making the instrument very sensitive and, from its small size and simplicity, very convenient.

The Barometer is used to detect changes in the pressure of the

atmosphere caused by the movement of the ocean of air which surrounds the earth. When the heat of the sun is concentrated more in one locality than another, the air becomes expanded. This expansion may go on until a reaction sets in. The atmosphere when clear allows the sun's heat to raise the temperature and generate moisture, which, rising to form clouds, obscures the sun and intercepts the rays, which in turn, together with the reaction, sets up currents which we call wind. These currents, if from cold climates, condense, and at the same time drive the atmosphere, like a tidal wave, into one locality more than another, changing the pressure. The use of a barometer is to indicate these changes.

The range of the barometer is from about 28½ to 31 inches of mercury column, or a change of pressure from 2014 to 2190 pounds per square foot.

In mines where gas is driven off at little or no pressure, it would seem reasonable to suppose that when the atmospheric pressure fell, more gas would issue from the coal, or in cases where gas might be standing in wastes that it would expand and mix with passing currents, especially when we consider that a fall of one inch is equal to a reduction of pressure amounting to 70.6752 pounds per square foot. It sometimes happens that a rise of temperature, as indicated by the thermometer, is accompanied by a reduction of pressure, as indicated by the barometer. In such cases, if a mine depends for ventilation on the action of a furnace, the increase of temperature in the downcast may be nearly one-half the difference of temperature produced in the upcast by the furnace, thus reducing the pressure produced nearly one-half and the current one-fourth, while at the same time the reduction of pressure was letting off more gas. This is what has led to the use of the barometer in mining, especially with furnace ventilation.

A mercurial column over 30 inches in length is the most reliable to hang on a wall; but it is not portable, and has led to the introduction of the *aneroid barometer*, Fig. 13, which is a small brass drum exhausted of air, and, as the ends are corrugated, they act as springs and respond to every change of pressure. The motion of the ends is transmitted and amplified by levers and

gearing, so that a pointer marks the changes on the dial, which is graduated to correspond to a column of mercury, the instrument being made and graduated from comparison. These barometers can be used to measure the vertical depth of mines or height of mountains approximately, a movement of the pointer corresponding to one inch representing a change of elevation of about 900 feet.

The Thermometer is an instrument for measuring variations of temperature, and consists of a small glass tube terminating at the bottom in a bulb. The tube is partly filled with mercury and exhausted of air. When the temperature increases it expands the mercury and forces it upwards in the tube, and as the section of the bulb is large in comparison to the section of the tube, a slight increase of bulk in the former causes a large movement in the latter. Where furnace ventilation is used, the thermometer is required to ascertain the difference in temperature between the air entering the mine and that in the upcast as produced by the furnace, so as to note any change or to calculate the efficiency of the furnace or the power it is exerting.

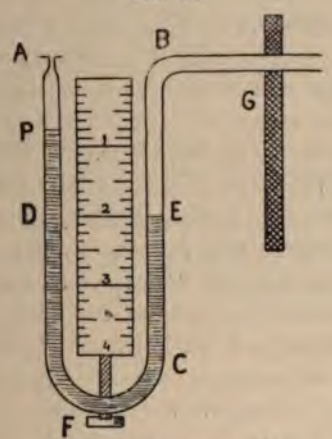
The Hygrometer is a combination instrument for determining the amount or proportion of moisture in the air of a mine, so as to ascertain its density and capacity for heat, but is only used for fine scientific experiments. There are several different kinds of this instrument used, but as experiments made with it involve complicated problems in hygrometry, it is seldom used in mining, and a description of it is, therefore, unnecessary. The fact that air saturated with vapor has less weight than dry air is of very little practical importance in a properly ventilated mine.

The Water-gauge is an instrument for measuring the pressure or force expended on every unit of surface of the area of section, to create the velocity, and overcome the resistance to currents of air in mines. It is simply a tube bent in the shape of the letter U, and may be best described by the aid of a diagram.

Let *A C B* (Fig. 14) be a bent tube of glass with parallel legs partly filled with water; the end *A* is open, and the part *B G* is bent at right angles for convenience in placing the instrument through a hole, so as to show the difference of pressure on op-

posite side of an air partition, etc. *F* is a thumb-screw for adjusting the scale to bring a division in line with the surface of the water in one of the tubes, and thus show the difference of level of both in inches and fractions. Sometimes the scale is fitted with a vernier. Let us suppose the end of the tube *G* to be inserted in an auger hole bored in the side of the air-drift which connects the mine with the fan, just sufficient distance back to be clear of the disturbance caused by the blades passing the discharge opening. Suppose the fan to be forcing or acting as a blower, then the pressure inside is greater than outside, and will force the water down in the leg next the fan and up

FIG. 14.



Water-gauge.

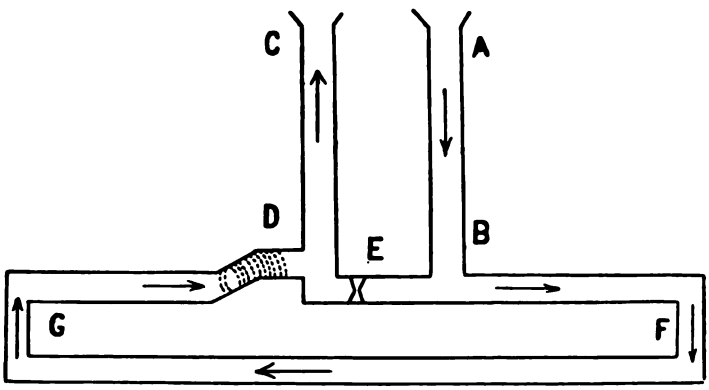
the other. Let the line *D E* be a horizontal plane touching the surface in *B C*, and *P D* the difference of level equal to one inch, as shown by the scale. Suppose, for simplicity, that the internal section of the tube is one inch square, then the pressure of the fan will be supporting a column of water equal to one inch high and one inch square, or one cubic inch; therefore, the fan is exerting a pressure in excess of the atmosphere equal to the weight of one cubic inch of water on every square inch of area of fan drift at that point, or 144 times the weight of a cubic inch of water in every square foot. As ventilating pressures

are very light, they are generally measured in square feet, and as a column of water a foot square and one inch deep weighs, at ordinary temperature, 5.2 pounds nearly, therefore the difference of level *P D*, equal to one inch, will represent a difference of pressure above the atmosphere of 5.2 pounds per square foot. If the difference of level was several inches, then the inches multiplied by 5.2 will give the pressure in pounds per square foot. If the one leg of the instrument was much larger than the other, it would not change the result, because the water in the small tube would move just as much further up or down as the difference in the sections of the tubes, and produce the same difference of level with the same pressure. If mercury was used instead of water, the pressure that equalled one inch of water would only show 1/13½ of an inch of mercury, it being 13½ times heavier. The scale for mercury would be so small that it would be difficult to read fractions of pounds in the pressure. This is the reason that mercury is not used to measure ventilating pressure. The only difference with an exhaust fan would be to raise the water in the opposite side of the gauge.

When the water-gauge is applied at a fan it shows the pressure necessary to overcome the resistance of the air in moving through the mine, and also the force required to generate the velocity, because it is measuring at the point where the force is applied, and comparing it with the atmosphere where there is no resistance to overcome or velocity to generate. But when applied, as below, to measure the pressure exerted by a furnace at the bottom of a shaft, it only shows the pressure spent in moving the air from the bottom of the downcast around the workings and back to the furnace, and does not show the pressure spent in overcoming the resistance of the shafts, which, when deep, may be nearly one-half of the whole resistance. In order to make this more easily understood, in the accompanying cut, Fig. 15, let *A B* and *C D* be a downcast and upcast shaft respectively; let a furnace placed at *D* be causing a current of air to circulate, as indicated by the arrows, down one shaft, around by *F* and *G*, through the furnace and up the other shaft. If the water-gauge is placed in a door or partition which separates the two shafts at *E*, it will show the pressure which is for-

cing the air from *B* round by *F* to *G* and *D*, at the furnace, where the force is being applied; but it does not measure the pressure spent from *A* to *B* in the downcast, or from *D* to *C* in the upcast, because if the partition at *E* was removed, the air would

FIG. 15.



rush through to the shaft *C D;* therefore, the pressure is the force necessary to drive the air round by *F G*, and the height of water column maintained in the gauge at *E* is the measure of that pressure.

CHAPTER III.

VENTILATION.

Ques. 1.—State your views on the importance of ventilation as a sanitary measure.

Ans.—There is no subject of more importance in mining or more worthy the attention of mining students than that of ventilation.

If the machinery for raising the water from a mine has not sufficient power to perform the work, the water will accumulate and the mine will be flooded. A given weight of water has to be lifted a certain height in a certain time, which requires the expenditure of so much power. And the management must either furnish the power or the mine must stop; there can be no compromise effected with water. If the power employed to move coal in the mine and hoist it to the surface be inadequate, the number of tons expected will not reach the market, and, as the amount of profit is directly dependent on the quantity marketed, the management must do one of two things, either increase the hauling and hoisting capacity or lose the tonnage.

With ventilation the case is a little different.

Men can do some work in a poorly ventilated mine. Inadequate ventilation does not peremptorily stop operations like water; therefore, the question of ventilation has generally been left to take care of itself, except where danger from gas steps in and compels at least some attention to the matter. The fact that even now our best ventilated mines are those which give off dangerous volumes of gas, shows that the modern improvements in ventilation are due more to a dread of accident from explosion than to a consideration of the health or comfort of those employed. Not many years ago it required the force of public opinion, through the power of legislation, to effect improvement in ventilation.

Happily for all concerned, a brighter day has dawned, and more attention is being directed to this important branch of mine management. The fact is impressing itself on the minds of those who are pecuniarily interested in coal mines that no part of the capital invested yields a higher rate of interest than that which is judiciously expended on ventilation.

It would require a long time, much labor, and large space to set forth properly the reason of its great importance as a sanitary measure, as a matter of economy, and as a safety appliance.

In order to appreciate fully the difference between good and bad ventilation in its effect on man's health, we ought, at least, to know something about the reason *why* we breathe the air of heaven so as to live, move, and have our being.

From our heart, and extending to every part of our body, is a wonderful and complete system of tubes for the conveyance of the blood, one set to carry it from, and another to return it to, the heart. This heart is the pump which forces the blood through these "pipe-lines," and the pulse at the wrist marks the number of strokes per minute.

Human blood under the microscope presents two separate and distinct parts; one a thin and nearly colorless fluid, the other a myriad of minute disks floating in its current. The diameter of these disks is from $\frac{1}{2800}$ to $\frac{1}{4000}$ of an inch, so that the blood, as it is forced through the *arteries* to every part of the body, contains millions of minute vessels floating in its current, laden with material to build up and keep in repair this wonderful structure, the body, and returning by the *veins* with its stream polluted by the refuse, in black carbonic acid set free in the tissues from combustion of the nutriment by the action of the oxygen in the blood.

When we breathe, the cells of our lungs are filled with air. Here the blood absorbs the oxygen and bears it to all parts of the body, depositing it where most required.

Laden with this life-sustaining element, the vital fluid sweeps tingling through every artery, distends every capillary tube, sending strength to the nerves and a flush to the cheek; combines with the food thrown into the circulation from the stomach, breaks up every worn-out tissue, and sets free their force.

After having run its course and performed its mission, it returns back through the veins, dark and thick with the products of combustion, the cinder and ashes of the fire within us—this human furnace.

When the blood returns it passes through the innumerable air-chambers of the lungs, where the carbonic acid gas is thrown off by the breath, and freshly charged with oxygen from the air, forced by the heart into the arteries, it goes off again, and so on the current courses, and any cause that stops the stream produces *death*. The quantity of oxygen taken in and the quantity of carbonic acid discharged bear a very regular proportion to the amount of exertion undergone by the individual. He cannot lift a finger or move a muscle without a corresponding expenditure of force, and the faster the motion and the more work performed in a given time, the quicker the wear and the more new material and fuel required to make up the waste and keep up the stream; the little disks must float faster and carry more, must come back sooner and be emptied quicker, which can only be accomplished by a more abundant supply of pure fresh air.

On an average, about 360 cubic feet pass through the lungs in twenty-four hours, or about 15 cubic feet per hour, and, as that which has *once* passed through them contains $\frac{1}{24}$ of carbonic acid, it follows that about 15 feet of this gas, containing nearly 8 ounces of solid carbon, are thrown off in the course of twenty-four hours.

This carbonic acid when diffused to any extent is most injurious. The usual proportion present in the atmosphere is about 5 volumes in 10,000, but if this is increased to $\frac{1}{100}$, its effect begins to be felt in headache, languor, and depression. The poisonous effect increases in intensity with the percentage, until a point is reached producing suffocation and death. Why? For the very same reason that the colliery must stop when the chutes are blocked. The blood current is choked and circulation ceases.

This is a sketch of why we breathe to live, showing that we must have air even if we do not move, and must have more if we work. This being the imperative demand of nature on the surface, how much more should it be in the dark chambers of the mine?

When we take into consideration the different causes at work tending to vitiate the atmosphere of our mines, we begin to realize the importance of supplying them with currents of fresh air.

If we take one of the collieries in the Shenandoah, Pa., district as an example, we find from the inspector's report that 325 men and boys are employed inside, and from 25 to 30 mules, and that 20 kegs or 500 pounds of powder are burned for every day the colliery is in operation.

Now if we consider a mule equal to seven men, and every lamp equal to a man, we will have as follows:—

Men and boys,	325
Mules, $30 \times 7 =$	210
Lamps,	325
Total,	860 men.

It has been shown that about 15 cubic feet of CO_2 gas are thrown off by each man in 24 hours. If we multiply our 860 men by 15, it will show what would be the average for the whole:

$$\frac{860 \times 15}{24} = 538$$

cubic feet per hour, and

$$\frac{538}{60} = 8.9$$

or nearly 9 cubic feet per minute of carbonic acid gas thrown off merely from respiration.

This would cause headache in a current of 900 cubic feet on the surface, but when we consider that man throws off nearly as much more through the skin, bowels and kidneys, the effect in the mine is doubled.

The explosion of this 500 pounds of powder takes place during the day, and may be considered as in 10 hours, or 50 pounds per hour, or nearly one pound per minute. The Anthracite Mine Law calls for 200 cubic feet of air per minute for each employé, as the minimum, and as much more as may be necessary, which for the above-mentioned colliery would be

$$325 \times 200 = 65{,}000 \text{ cubic feet per minute.}$$

When we consider the volume of powder smoke to be cleared away, the gas given off from the strata, the carbonic acid produced from decaying timber and other organic matters, together with the vitiation produced by the breathing of men and animals before noticed, we begin to realize the meaning and design of the law. And there is no practical difficulty in doubling the quantity if required.

Although the actual quantity required in a colliery is dependent on circumstances, still the above may help us to view the matter from a practical standpoint and lead to reflection on this important subject.

Ques. 2.—What are the natural laws affecting the ventilation of mines?

Ans.—The physical properties of air and gas, which may be briefly noticed as follows:

First. Air is impenetrable, the same as solids or fluids; that is, air and something else cannot occupy the same space at the same time. If we pour water into an open vessel, the air will be pushed out. If the vessel be closed and the air cannot escape, the water will have to be forced in and the air will be compressed into as much less bulk as the space occupied by the water.

Second. Air, like other matter, is affected by gravity, and, therefore, has weight. When the pressure of the air, as shown by the barometer, is 30 inches and the temperature is 32°, or at the freezing-point, a box which would contain 13 cubic feet would weigh 1.0526 pounds less if exhausted of air.

Third. Air, having weight, it must necessarily have inertia. It requires force to put it in motion, and when in motion it must meet with some resistance to bring it back to rest. Its inertia is exactly proportionate to its weight, which, in proportion to its volume, is so small that a very small force is sufficient to put a very large volume of air in motion.

Fourth. Air exerts pressure because it has weight, and the space it occupies, or the volume it assumes, is inversely proportional to that pressure. The atmosphere which surrounds the earth being elastic, its density is greatest at the lowest level, and the sea-level being the natural bottom, it is used as the

basis in calculations. If we go up a mountain, we find less pressure, because there is less height or head of air above us, and what air is above us has less weight in proportion to its bulk, because it is less and less compressed by its own weight as we ascend, until it becomes so rarefied as to be unfit to support life. If we go down a shaft below the sea-level, we get more density, because we have the usual pressure at sea-level with that due to the depth below that level added. When being lowered rapidly in a deep shaft one can feel this increase of pressure on the drum of the ear, the sensation being caused by the rapid descent into a denser atmosphere. If we take a glass tube about three feet in length, tightly closed at one end, and fill it with mercury, then close it tight with the finger and invert it into a shallow cup containing just sufficient mercury to cover effectually the end of the tube and prevent air from entering, remove the finger and the mercury will run down in the tube until its height is between 28 and 30 inches, when, after a few oscillations, it will come to rest. The reason why it does not all run down into the cup is that when it was inverted it was full, and when the mercury ran down to about 30 inches the space above it was left as complete a vacuum as can be produced by mechanical means. Therefore the pressure of the air cannot act on the surface of the mercury in the tube, but it acts on the surface of the mercury in the cup, and this pressure transmitted to the mercury maintains it at that height. This 28 to 30 inches of mercury, therefore, balances the weight of a column of air having the same section and the full height of the atmosphere. If the tube has an internal section of one square inch and the mercury stands 30 inches above the level of the mercury in the cup, and as a cubic inch of mercury at the usual temperature weighs 0.4908 of a pound, therefore

$$30 \times 0.4908 = 14.724$$

pounds pressure per square inch is being exerted by the weight of the atmosphere. This tube is a barometer. If we do the same with a long tube filled with water, it will stand at about 33 feet, water being so much lighter than mercury. It is by this pressure that a boy lifts small stones with a leather disc and

a string; that a siphon pipe lifts water to a height 30 feet above its source; and that water rises in the suction pipe of a pump.

Fifth. Air is elastic, and can be compressed to half its bulk by doubling the pressure to one-fourth of its volume by four times the pressure, and so on. If we invert a tumbler upon the surface of water and push it down about 33 feet, it will be only half full, but will still contain the same weight of air, because at the surface it was pressed by a weight equal to about 33 feet of water column, and when down in the water 33 feet the pressure is doubled. To illustrate this principle, we may compare the atmosphere to a spiral spring, supposing the spring to be 100 feet in perpendicular height and weighing one pound per foot, or 100 pounds. The bottom coil would be pressed by a weight of 100 pounds, the middle coil by 50 pounds, and the top coil by only its own weight. If the bottom coil was squeezed to one inch by the 100 pounds, the middle coil would be squeezed to 2 inches by the fifty pounds, and so on up until the last coil is not affected except by its own weight. Air is a perfect spring and, when compressed, will give back all the force put into it when the pressure is removed. The bulk of a given weight of air will, therefore, vary as the pressure when the temperature remains constant.

Sixth. Air, like other gaseous bodies, expands with heat; a rise of temperature increasing its volume if the pressure remains constant. Thus a bladder filled with air bursts when heated, and a furnace or stove causes a draft through itself by the heat rarefying the air and making it lighter, and, the surrounding air being heavier, pushes or rather buoys the light air upwards. Careful experiments show that air expands $\frac{1}{459}$th of its volume at zero degrees Fahrenheit for every degree added to its temperature when the pressure is constant, therefore 459 cubic feet at zero will become 460 feet at 1°, 469 feet at 10°, and so on, as long as the pressure remains unchanged. The fact that the *bulk* for a given *weight*, and, of course, the *weight* for a given *bulk*, varies, the one inversely and the other directly as the pressure, together with the fact that it expands $\frac{1}{459}$ of its volume for every degree of heat added under constant pressure, are two very important properties of air in regard to the natural laws

affecting ventilation of mines. These properties of air, together with the law of the diffusion of gases already noticed, and also the theory that a reduction of atmospheric pressure allows an increased quantity of gas to escape from the coal and fissures in the strata, are the principal causes affecting mine ventilation.

Ques. 3.—Explain the theory of natural ventilation, and describe its effects "practically" in mines dependent on natural causes for ventilation.

Ans.—In order to understand the cause of natural ventilation we must remember that the height of the atmosphere does not

Fig. 16.

follow the undulations of the earth's surface, so that if it is 30,000 feet high from the bottom of a valley, it will only be 29,000 feet above the top of a neighboring mountain whose summit is 1000 feet above the valley.

To illustrate natural ventilation, let Fig. 16 represent the profile of a mine entering the side of a hill by a water-level tunnel. Let *A D* be the contour of the hill, *B C* a seam of coal, *A B* the tunnel cutting the coal-seam at *B*, and *B C* an airway, at the upper end of which the air-shaft *D C* is sunk and connected to the workings. Let the dotted line *A E* represent an imaginary column of air whose top is on a level with the top of the air-

shaft, and the dotted line *D F* an imaginary continuation of the air-shaft down to the level of the tunnel *A B*.

Now it is evident that if the air in the mine from *A* to *B*, from *B* to *C*, and from *C* up to *D*, is of the same temperature as the air outside from *A* up to *E*, there will be no difference in weight, and the air in the shaft will balance the imaginary column *AE*, and there will be no motion in either direction. The temperature inside does not change as much as the temperature outside. In winter the air outside is much colder than inside, and the air from *B* up to *D* is lighter, bulk for bulk, than the air from *A* up to *E*.

Therefore, the cold and heavy air flows in at *A* with a force proportionate to the difference of weight between the cold and the hot air. This drives the light air up the shaft *CD*. The cold air flowing into the tunnel, by coming in contact with the warmer surfaces in the mine, is heated and expanded in its turn, and, as it cannot back out, against the denser air, it seeks the line of least resistance up the air-shaft and out at *D*. In summer this is reversed. As the warm light air on the hill pours into the shaft *DC*, it is cooled by coming in contact with the cold surfaces in the mine, and, becoming denser and heavier, it flows out at the mouth of the tunnel *A* and displaces the lighter air outside, until, by mixing, it is heated and expanded to the same weight as the imaginary column *AE*. Oftentimes in calm weather the cold current can be felt flowing out along the ground 60 to 100 feet, and in winter the water dripping from the top and sides is formed into columns of ice three or four hundred yards in from the entrance.

This is the natural ventilation which supplied our mines with air before the introduction of furnaces, steam jets, fans, etc. In the heat of summer and the cold of winter this power moved limited quantities of air through the mines, but a time came in spring and autumn when the temperature outside and inside was about equal, and, as this made the two columns of air nearly the same weight, there was no ventilation.

To describe the practical effect of this system of ventilation, and to appreciate the actual condition of some mines under this system a few years ago, let us suppose the mine represented by

Fig. 16 to be doing a small business, having say 8 chambers and 2 gangways, employing 20-odd miners blasting coal.

During the summer, when the demand for coal was slack, the air in the mine would be sufficient; but when autumn came, and the demand for coal became brisk, the air became so poor that the men could work only a few hours a day, the miner lost time, and the owner lost trade; the one was ruined physically and the other financially, for the want of circulating the free air of heaven through the mine.

In the cold autumn mornings, when the men started work, the air was moving in at *A* and up the air-shaft. Between 8 and 9 o'clock A. M. the first round of shots would be fired, and, as the sun had by that time raised the outside temperature nearly equal to the inside, the current became very slow. The chambers were now filled with a cloud of powder smoke, and the miners would go down on the gangway until it cleared. In half an hour, perhaps, objects could be dimly seen at the face, and the men would resume work.

In the meantime, the column of powder smoke had moved off towards the air-shaft, and by the time the front end of the column had reached the shaft the current had become imperceptible. It might now be near 10 o'clock and another round of shots would be fired. The men would go down on the gangway to lunch. In about an hour they would go up to see if the smoke had cleared. They found it only diffused, and still a cloud so dense that neither top nor sides could be seen. It being nearly noon, the sun had made the outside and inside temperature equal, and the ventilation had stopped. The men could do one of two things, either go home and lose their work, or persevere and contract a headache for the remainder of the day. After waiting themselves out of patience, they would grope their way up the chambers, feel for their tools and a place to use them, and start to work. After groping, stumbling and trying to work amid the smoke for an hour or two, the chamber had not cleared, but cooled a little. Just as objects were beginning to be dimly visible, the current began to reverse, and crawl backwards towards the mouth of the tunnel, the sun having raised the outside temperature above that of the mine.

This change of direction began to drag the column of smoke that had moved towards the air-shaft back over the men, and as the day wore on it began to move a little faster, and the whole column of smoke, which filled the mine from the chambers to the air-shaft, was dragged slowly but surely back over the very place where it was produced.

By the time the shaft end of the column had reached nearly to the men, and all between them and the entrance of the tunnel filled with smoke, the current again stopped, preparatory to its evening performance in the other direction, the temperature outside having begun to lower in the afternoon. By the time it got started to travel back over the men for the second trip, they were fairly "knocked out," and scrambling down to the gangway, groped around for their dinner cans, and started for the mouth of the tunnel, staggering and trembling in every limb, only getting clear of their deadly foe when they reached the open air. After sitting in the little uncomfortable shed at the entrance to recruit strength to move on, they stagger home as best they can, weak, dizzy and confused, and feeling as if a metal hoop had been shrunk around their temples. When the men were fairly conquered and complained, the purely practical boss of the period would ask if they expected him to give them air when there was none outside, and so they could go home and wait until the weather made some.

This is no exaggerated picture. We have plenty of old miners around to convince the gainsayers.

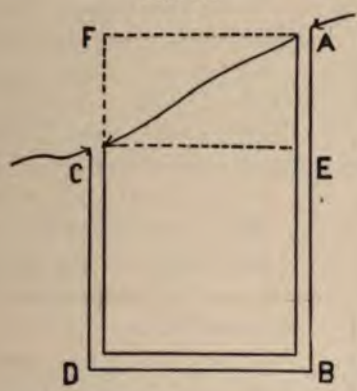
Natural ventilation in mines opened by shafts or slopes is still worse than with mines above water-level, because their difference of surface level is generally so little in comparison to their depth that the pressure due to difference of temperature is unable to overcome the friction. In deep shafts, if the inside temperature is very high, when the current is once started the upcast becomes warm and the downcast correspondingly cold, but a warm day may heat the downcast and stop the current.

For the purpose of illustration, let Fig. 17 be two shafts sunk from different surface levels. If A in the shaft AB was 50 feet higher than C in the shaft CD, then the difference of weight between the 50 feet from E up to A, and the 50 feet of imaginary

column of outside air from C up to F, would be the pressure producing the current.

Because 50 feet are contained 9 times in 459, and because 459 feet of air expand one foot for every degree of heat added, it would require a rise of 9° to expand or contract the column one foot and produce a pressure equal to the weight of a cubic foot of air. Therefore, any difference of temperature produced under usual conditions would not be of any practical use for ventilation. By neglecting all below the level of $C\,E$, the same reasoning will apply in this case as with the mine above water-level. The amount of pressure is measured by the difference of temperature and height between the openings of the mine.

FIG. 17.



It is only during very cold weather that natural ventilation can be of any practical use.

Take a colliery 1000 feet deep, worked either by shafts or slopes, under the most favorable conditions; if the temperature on the surface was nearly at zero, and the cold air reaching the bottom of the shaft or slope even at the temperature of 7° or 8° below freezing, and the mine and upcast, say 65°, which is rather high for any mine even at that depth, then the ventilating pressure under these conditions would be as follows:—

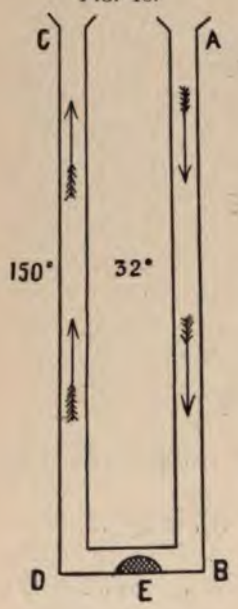
1 cubic foot of air at 25° weighs 0.0821460 of a pound,
" " " 65° " 0.0758753 " "

Difference of weight per foot is 0.0062707 of a pound.

Then, $\frac{0.0062707 \times 1000}{5.2} = 1.2$

inches of water-gauge. This might produce a fair ventilation while the severe frost lasted, but whenever the thaw came the ventilation would cease. There are few, if any, collieries where this result would be reached, and the average mine would not produce over half the above pressure.

Fig. 18.



Ques. 4.—What means are employed to produce ventilation in mines?

Ans.—Furnaces, fans, displacement machines, steam jets, the exhaust from underground steam engines, and, sometimes, in case of accident, compressed air. Falls of water have been used in shafts to start a current after an accident.

Ques. 5.—Describe furnace ventilation.

Ans.—Furnace, like natural ventilation, being the result of *rarefaction* by increase of temperature, is governed by the same laws.

If two shafts are sunk from the same level, and connected at the bottom, by placing a furnace near the bottom of one shaft, it rarefies the air. The heavier air in the other shaft buoys it upwards, the cold air being heated in its turn as it passes through the furnace, and so on; as long as the furnace can impart more heat than the sides of the shaft can absorb, the current will continue to move. Furnaces placed in the bottom of deep dry shafts will induce large currents of air, but in a climate where the surface temperature ranges from below zero to ninety above it, their action is very variable.

To illustrate how furnace ventilation is affected by change of

season or weather, let Fig. 18 be two shafts, 1000 feet deep, connected with extensive workings. In winter the temperature in the downcast *A B* may be 32°. Suppose the furnace *E* raises the temperature in the upcast *D C* to 200°. It is found that in dry furnace shafts, where the speed is about 800 per minute, the air loses a degree for every ten feet of ascent. This would be a reduction of $1000 \div 10 = 100°$ when the air reached the top, then the average temperature in the upcast would be 150°. If the barometer half way down the shaft registered 30 inches, then the weight of a cubic foot of air in the downcast would weigh

$$W = \frac{1.3253 \times 30}{459 + 32^{\circ}} = 0.08097 \text{ lb.}$$

and a cubic foot in the upcast would weigh

$$W = \frac{1.3253 \times 30}{459 + 150^{\circ}} = 0.06528 \text{ lb.}$$

the difference of weight per cubic foot would be

$$0.08097 - 0.06528 = 0.01569 \text{ lb.}$$

The shafts being 1000 feet, the total pressure per foot moving the current would be

$$0.01569 \times 1000 = 15.69 \text{ lbs.,}$$

equal to a water-gauge of

$$\frac{15.69}{5.2} = 3.02 \text{ inches.}$$

In summer, with the temperature on the surface at 90° and the average in the downcast 75°, the weight per cubic foot would be

$$0.07445 - 0.06528 = 0.00917 \text{ lb.}$$

$$\frac{0.00917 \times 1000}{5.2} = 1.76 \text{ W. G.}$$

Because the ventilation varies as the square root of the water-gauge, the differences between summer and winter under these conditions would be $\sqrt{3.02} = 1.73$ and $\sqrt{1.76} = 1.32$ and

$$\frac{1.73}{1.32} = 1.31.$$

So that the difference between the volume of air in summer and winter would be as 1 is to 1.31 or a third more air.

The air being expanded in the upcast shaft, we would suppose it should be the larger, so as to keep down the speed of the current; but it must be remembered that increased area also increases the cooling surface, and, by slowing the speed, gives the air longer time in contact with it, and, as the expanded air has less friction for the same bulk, it ascends faster with the same pressure. If the shaft is wet and absorbs much heat, it should not be too wide; if lined and dry, then a large area would be an advantage.

Ques. 6.—Explain the formula for finding the weight of air per cubic foot at different temperatures and under different pressures.

Ans.—The formula as given to find the weight of air is

$$W = \frac{1.3253 \times B}{T + 459}.$$

Where W = weight in fractions of a pound per cubic foot; B = height of barometer in inches of mercury column; T = temperature by Fahrenheit's thermometer.

As previously stated, air expands $\frac{1}{459}$ of its volume for every degree of heat added under constant pressure, 459 cubic feet at 0° will become 559 at 100°, and so on; therefore, if we add the constant 559 to the temperature of air, the sum will represent the bulk at that temperature as compared with its bulk at zero. Because if 459 cubic feet or inches be raised in temperature 100°, its bulk will have increased $\frac{100}{459}$ths, which will be 100 + 459 = 559. Its bulk having increased, it must be remembered that its weight per foot has decreased in the same proportion.

It has also been stated that air expands or contracts in direct proportion to the pressure when the temperature remains constant. Therefore the bulk, and, of course, the weight per cubic foot will vary directly with the height of the mercury column as indicated by the barometer.

Careful experiment shows that when the barometer stands at 30 inches with the mercury at a density due to 32°, or freezing, 459 cubic feet of air at zero weigh 39.7587 pounds; therefore, if all the pressure was removed except that due to one inch of mercury, it would expand to 30 times the bulk, and 459 feet of

the expanded air would then weigh $\frac{1}{.8}$ or 1.3253 pounds per cubic foot. In the formula given,

$$1.3253 \times B$$

means that 1.3253 multiplied by say 30 inches of mercurial column, would be equal to

$$1.3253 \times 30 = 39.759$$

pounds as the weight of 459 cubic feet at that pressure.

If we now divide this by 459 cubic feet at 0° it gives the weight of one cubic foot at zero when under that pressure, or

$$\frac{39.759}{0° + 459} = 0.0866 +$$

of a pound per cubic foot.

But suppose we want to find its weight at 100°, then because the air at 100° would be as much lighter as its bulk was greater at that temperature, if we add 459 to 100 the sum 559 will be the proportional bulk to the original bulk at zero, and, of course, if we divide the weight 39.759 by the bulk at the increased temperature it will give the weight per foot at 100°.

This would just be the given formula,

$$W = \frac{1.3253 \times 30}{459 + 100°} = \frac{39.7587}{559} = 0.07112 +$$

of a pound per cubic foot.

In the above formula the number 39.759 is the weight of the given bulk of 459 feet, and the number 559 is the increased bulk of that same weight, and the quotient .07112 is the weight per unit of bulk.

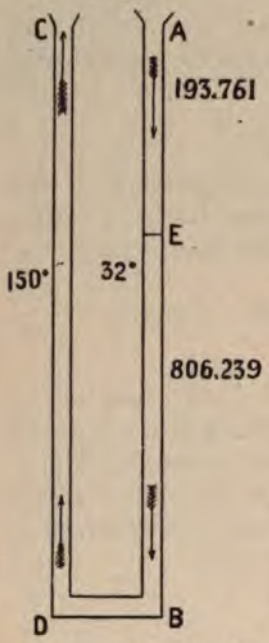
Ques. 7.—Explain what is meant by the term "Motive Column."

Ans.—The "Motive Column" is the length of a column of air in the downcast of such length that its weight is equal to the difference of weight between the air in the down and upcast shafts. Its weight is, therefore, the "pressure" producing ventilation, and when its length is reduced to an equivalent column of water, it is the water-gauge.

The formula to find the motive column is

$$M = D \times \frac{T - t}{T + 459}$$

Fig. 19.



Where M = motive column in feet, D = depth of shaft in feet, T = temperature of upcast shaft, and t = temperature of downcast shaft.

In order to investigate the formula given, in Fig. 19 let *AB* and *CD* be two shafts 1000 feet deep, the temperature of *AB* and *CD* being 32° and 150° respectively.

If the column *AE* be taken from *AB*, leaving *EB* = in weight to *CD*, then *AE* is the motive column. The weights of *EB* and *CD* being equal, if we add to their temperatures the constant 459, the sums will be proportional to their respective lengths, and as the length of *CD* is known, the length of *EB* can be found by proportion.

$$150° + 459 = 609 \text{ and } 32° + 459 = 491.$$

Then

$$609 : 491 :: 1000 : 806.239 \text{ feet} = EB,$$

and

$$1000 - 806.239 = 193.761 \text{ feet} = AE = M.$$

This is an indirect method, because we have first to find *EB* and subtract it from *AB* to find *AE*.

Now because the lengths of *CD* and *EB* are proportional to their temperatures plus 459, their difference of length *AE* will be to *CD* as the difference of their temperatures is to the temperature of *CD* plus 459, or

$$609 : 118 :: 1000 : 193.761 \text{ feet} = AE = M.$$

This is just the formula as given.

$T + 459 : T - t :: 1000 : 193.761$ feet $= AE = M = 1000 \times \frac{118}{609} = 193.761 = D \times \frac{T - t}{T + 459} = 193.761 = AE = M.$

This being the length of column equal in weight to the pressure. If we consider the space AE a vacuum, then a weight falling that distance would acquire a velocity equal to 8 times the square root of the height $E\,A$, $\sqrt{193.761} = 13.91 \times 8 = 111.28$ feet per second; $111.28 \times 60 = 6676.8$ feet per minute as the velocity which the motive column would attain by falling through its own length, which is the theoretical velocity of the air produced by the pressure or weight of the motive column *A E, if there was no resistance.* Now, if it is found in practice that the air as it reaches the top of the upcast *CD* had a speed of only 1000 feet per minute as a final velocity, then we would know that $6676.8 - 1000 = 5676.8$ feet per minute of the speed had been lost by the air passing through the mine from meeting frictional and other resistance, the proportion lost being about

$$\frac{5676.8}{6676.8} = 85 \text{ per cent.}$$

showing that 85 per cent. of the velocity was spent on resistance in the mine. The pressure could be found as well in the upcast as the downcast, but as the meaning of the motive column is to find the motive force and theoretical velocity, it is measured by the head of air in the downcast equal to the pressure. Assuming water to be 833 times heavier than air, this motive column of 193.761 feet would be equal to

$$\frac{193.761 \times 12}{833} = 2.79$$

inches of water-gauge.

A velocity of a thousand feet in a $10' \times 10'$ shaft would be a volume of 100,000 cubic feet per minute. The above example is within the limits of actual practice where the shafts are deep, the workings extensive, and the mine well ventilated.

Ques. 8.—What is meant by the term "co-efficient of friction," and how can it be determined experimentally?

Ans.—When a current of air is forced through a tube or con-

fined passage, as in a mine, it requires a great deal more force or pressure to move it than would be necessary to produce the same speed in the open air.

A current of air or "wind" on the surface requires only force to produce the speed, while in a mine, in addition to the force necessary to produce the velocity, it may require 10 times more to overcome the resistance encountered by the air rubbing against the surface of the passages through which it moves and in force spent on changing its direction at bends and angles in its course.

This resistance is the friction, and the coefficient is the unit of pressure in fractions of a pound per square foot required to move the air over a unit of surface, or one square foot in a unit of time, or at a speed of one foot per minute.

The following symbols are generally used to denote the different factors necessary to work problems in ventilation, for the purpose of studying the subject:—

a denotes the area of airway in square feet.
l " " length of airway in lineal feet.
o " " perimeter of airway in lineal feet.
s " " surface " " square feet.
p " " pressure in pounds per square foot of section.
k " " co-efficient of friction in pounds per square foot of surface rubbed.
q " " quantity of air in cubic feet per minute.
v " " velocity of air in feet per minute.
u " " units of work or foot-pounds expended.

The value of k, the co-efficient of friction, is found by the formula

$$k = \frac{p\ a}{s\ v^2}$$

For the purpose of reducing this formula to words, let us suppose, for example, an airway 5×5 and 1000 feet long, with a current of air passing through at a speed of 500 feet per minute. under a pressure of 4.34 pounds per square foot.

$5 \times 5 = 25$ = a the area.
$4 \times 5 = 20$ = o " perimeter.
$20 \times 1000 = 20{,}000$ = s " surface.
$500^2 = 250{,}000$ = v^2 " the velocity squared.
4.34 pounds = p " the ventilating pressure.

Substituting these known quantities in

$$k = \frac{p\,a}{s\,v^2}$$

it becomes

$$k = \frac{4.34 \times 25}{20{,}000 \times 250{,}000} = 0.0000000217$$

of a pound, being the coefficient generally used in the text-books. This coefficient was determined by Mr. J. J. Atkinson in England, and is therefore known as "Atkinson's Coefficient." If the velocity was one foot per minute, the quantities in the formula would become

$$\frac{0.00001736 \times 25}{20{,}000 \times 1^2}\ 0.0000000217.$$

To double the speed to 2 feet per minute would require 4 times the pressure for every foot of surface; 3 times the speed would require 9 times the pressure, and so on. This is the reason that the *square* instead of the velocity itself is used as a factor. The velocity given in this example being 500 feet per minute, it requires 500^2 times the pressure for every foot of surface as a total pressure, which, divided by the area, gives the pressure per foot, or

$$\frac{(500^2 \times 0.0000000217) \times 20{,}000}{25} = 4.34$$

pounds, the given pressure. The formula given to find the pressure is

$$p = \frac{k\,s\,v^2}{a}$$

the same as above. In making calculations it is necessary to have a unit of measurement, and the coefficient of friction is that unit.

The value of the coefficient of friction is found by experiment,

because it varies according as the surfaces are rough or smooth. The friction in clear sheet-iron tubes is less than in wooden pipes, and much greater in the passage of a mine than in either. The value 0.0000000217 of a pound cannot be rigidly correct, because the friction will vary in different mines according as the sides of the passages may be dressed off or left rough, with or without timbers. Again, if the friction in different mines is equal within practical limits, the exact amount of rubbing surface cannot be measured, therefore calculations of the pressure necessary to pass a given quantity of air through a mine are only approximately correct.

If we wish to determine the coefficient of friction experimentally, let us suppose the airway given in this example to be nearly uniform in section and free from sharp curves or angles. If we take 200, or any number of feet of gas-pipe, and lay it along the airway, with the end facing the current closed and the other end open, then, because the current could not get into the tube, the pressure inside of the tube would be equal to the pressure at the open end. If a water-gauge was inserted through the closed end of the tube it would show the difference of pressure between the two ends.

The difference of pressure is the amount of force spent on friction from the air moving that distance. Then the pressure, as shown by the water-gauge, multiplied by the area or the total pressure, divided by the number of square feet of surface in the airway, for the length of the pipe, multiplied by the square of the speed of the air, will be the value of the coefficient of friction for that section of the airway, or

$$k = \frac{p\ a}{s\ v_2}.$$

If this was done for the gangways, airways, slopes, shafts, etc., with and without timber, an approximate value for the coefficient of friction in our mines could be found. The expense would be trifling, but I have seen no published report of such experiments being made in this country.

In Europe, however, more attention has been paid to this matter and numerous determinations of this coefficient have

been made. In the majority of cases the results are much less than the value given by Mr. Atkinson. Perhaps the most accurate and complete set of experiments are those made by Mr. D. Murgue in France. He experimented on rock-cut headings with bare walls, arched headings and timbered headings, and found that, of the three, the value of k is least in the arched headings and greatest in the timbered headings.

He also concludes that the value of k is greater in small sections than in larger ones and greater in curved or crooked headings than in straight ones. The following are some of his results:

The least and greatest values of k in five experiments on walled and arched headings are:

.0000000016.
.0000000033.

The section is 37 square feet in each case. The increase in the second case is accounted for by the heading being curved, while in the first case it was straight.

The least and greatest values of k found in four experiments on headings with bare sides are:

.0000000045.
.0000000064.

Both of these headings were straight, but in the first case the section was 45 square feet while in the second case it was only 22 square feet, which accounts for the difference.

Three experiments were made on headings timbered and lagged. The sets consist of a cap or cross-bar and two legs, and were placed about four feet apart, center to center. The least and greatest values of k are:

.0000000076.
.0000000126.

In the first case the heading was straight with a section of 36 square feet, and in the second case it was crooked, with a section of 22 square feet.

To make the meaning of Mr. Murgue's coefficients plainer, let us suppose we have a straight timbered heading 1000 feet

long with an area of 36 square feet = a and a perimeter of 24 feet = o. The surface of the air-way will then be $24 \times 1000 = 24{,}000$ square feet = s, and the pressure per square foot necessary to produce a velocity of 500 feet per minute would be

$$\frac{500^2 \times .0000000076 \times 24{,}000}{36} = 1.27 \text{ lbs.}$$

Now if the headings had the same sections and same surface but were arched instead of being timbered, the pressure per square foot required to put the same quantity of air in circulation would be

$$\frac{500^2 \times .0000000016 \times 24{,}000}{36} = .27 \text{ lb.}$$

or only a little more than $\frac{1}{5}$ of the pressure required by the timbered heading. If Mr. Atkinson's coefficient had been used, the result would have been much greater than either.

To the above figures for the friction should be added the force necessary to create the velocity as explained in answer to question 7. The friction and the force necessary to create the velocity are, therefore, the resistance in ventilation.

But because the force necessary to create the velocity is usually very small compared with the friction of the air in passing through the mine, it is usually omitted entirely in calculating the pressure required.

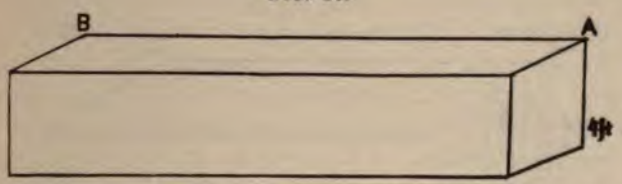
Ques. 9.—Explain the principles governing the friction of air moving through mines.

Ans.—The first principle to be noticed is "friction." Friction is produced by something rubbing against something else; and friction in ventilation is the result of the air current rubbing against the surfaces of the passages through which it is moving. When the barometer reads 30 inches the pressure on all surfaces exposed to the air is about 2120 25 pounds per square foot.

In Fig. 20 let *A B* represent an airway 4 feet wide and 4 feet high, and 1500 feet long. The amount of internal surface against which the air presses as it moves along is $4 \times 4 = 16$ square feet for every foot of its length, and as it is 1500 feet long the total surface is $16 \times 1500 = 24{,}000$ square feet. As the pressure per square foot, as before stated, is 2120.25 pounds, the

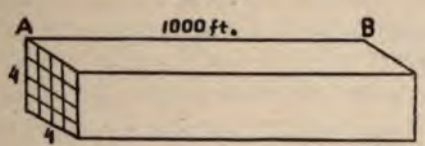
total pressure on this airway will be 24,000 × 2120.25 = 50,886,000 pounds, or 25,443 tons. If an airway only 4 × 4 and 1500 feet long bears a pressure of 25,443 tons, it is not difficult to realize the importance of friction. Friction being the result of this great pressure exerted by the air rubbing against the surface of the airway, it necessarily follows that it will increase or

FIG. 20.



decrease in proportion to the extent of the rubbing surface if the velocity of the air remain constant. Therefore, if the length of the airway is double, the resistance will be double and require double the total pressure to maintain the same velocity and quantity of air per minute. In ventilation, pressure as indicated by the water-gauge is measured in pounds per square foot of difference of pressure between the intake and return air, which difference is the pressure putting the air in motion. When an exhausting machine is used, it is a measure of how much less pressure is on the outlet than on the inlet; and when a forcing machine is used, of how much more pressure is on the

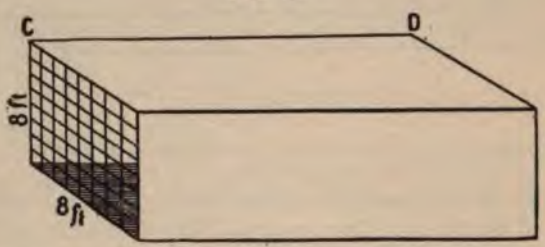
FIG. 21.



inlet than on the outlet. In the case of two airways of unequal area, but of equal rubbing surface, the smaller one has to be as much longer as will make up the difference. With the same total pressure the velocity in both will be equal, as both present the same resistance; but the same total pressure spread over the area of the large one will be as much less per foot as its area is

greater than the area of the small one, and as the velocity in both is equal, the volume passing in each will be in proportion to their areas. This principle may be illustrated by Figures 21 and 22. In Figure 21 let AB be an airway 4×4 and 1000 feet long, then the sum of its four sides is $4 \times 4 = 16$, and this multiplied by its length gives a surface of $16 \times 1000 = 16{,}000$ square feet.

FIG. 22.



In Figure 22 let the airway CD be 8×8 and 500 feet long, then the sum of its sides is $4 \times 8 = 32$, and this multiplied by its length is $32 \times 500 = 16{,}000$ square feet of surface. Their resisting surfaces being equal, they will require the same total pressure to produce the same velocity in each. If the pressure CD is 1 pound per foot, the total pressure will be 64 pounds. This on the area of AB would be

$$\frac{64}{16} = 4 \text{ pounds per square foot.}$$

Then because the area of CD is 4 times the area of AB, the speed being equal in both, if AB is passing 10,000 feet, CD will be passing 40,000 feet. Hence the large airway gives 4 times the ventilation with $\frac{1}{4}$ the water-gauge. If CD is made as long as AB, to maintain the same speed and continue passing 4 times as much air as AB, it will require $\frac{1}{2}$ the water-gauge of AB. This illustrates the advantage of large over small airways.

The second principle to be considered is the law governing friction at different velocities. If the pressure is increased, the speed of the air, and, of course, the quantity is increased, but not in the same proportion. Four times the pressure will produce only double the speed, nine times will increase the velocity

only three times, and so on in that proportion. *Therefore the quantity of air passing will vary as the square root of the pressure applied*, and the pressure will vary as the square of the velocity. If the pressure is reduced to one-fourth ($\frac{1}{4}$), the velocity will be still one-half ($\frac{1}{2}$) or the square root of $\frac{1}{4}$. If the pressure be changed to only one-ninth ($\frac{1}{9}$) the velocity will only be reduced to one-third ($\frac{1}{3}$), and so on in that proportion. It will assist us to understand the principle that the resistance of the air current in a mine and, of course, the pressure necessary to overcome it, *varies as the square of the velocity*, to imagine an air-way with something visible moving through it. Air being invisible, it is difficult to grasp a conception of its motion.

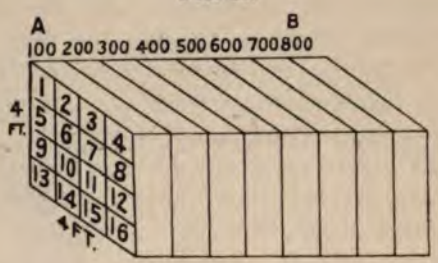
In Fig. 23 let *A B* be an airway $4 \times 4 = 16$ square feet area and 800 feet long, divided into 8 divisions of 100 feet each, with a column of steam moving through it at a velocity of one hundred (100) feet per minute. Suppose the pressure moving the column to be 2 pounds per square foot. Imagine the column of steam divided into blocks 1 cubic foot each. It is evident that these blocks, in moving, rub against the internal surface of the airway as follows:—

Block	1	has	2	sides	rubbing.
"	2	"	1	"	"
"	3	"	1	"	"
"	4	"	2	"	"
"	8	"	1	"	"
"	12	"	1	"	"
"	16	"	2	"	"
"	15	"	1	"	"
"	14	"	1	"	"
"	13	"	2	"	"
"	9	"	1	"	"
"	5	"	1	"	"
Total			16	"	"

Blocks 7, 6, 10, and 11, having no sides rubbing, create no friction. Each division of the airway will have $4 \times 4 = 16 \times 100 = 1600$ square feet of rubbing surface, and the whole air-

way will have $8 \times 1600 = 12{,}800$ square feet for the steam to rub against in passing through. The blocks, if moving at a velocity of 100 feet, will rub against 100 feet lineal or 1600 square feet per minute, and in eight minutes will rub against the whole 12,800 square feet in the airway. Suppose three times the quantity of steam is required, this will necessitate 3

FIG. 23.



times the velocity. If the blocks at a velocity of 100 feet per minute moved through one division and rubbed against 1600 square feet of surface, at 300 feet per minute they will move through 3 divisions and rub against $3 \times 1600 = 4800$ square feet. They will thus meet with 3 times the rubbing surface, which will cause three times the friction per minute, and if the pressure was 2 pounds per square foot at 100 feet, it will require $3 \times 2 = 6$ pounds at 300 feet per minute. But there is another cause of resistance which must be taken into consideration.

FIG. 24.

2	2	2	2
2	2	2	2
2	2	2	2
2	2	2	2

Total 32 pounds.

FIG. 25

6	6	6	6
6	6	6	6
6	6	6	6
6	6	6	6

Total 96 pounds.

The blocks, instead of striking against the frictional resistance with a momentum gained from a velocity of 100 feet per

minute, are now encountering the resistance with a momentum gained from a velocity of 300 feet per minute. This in itself creates a resistance three times as great, and requires three times the pressure to overcome it. This again increases the increased pressure three times and raises it to $3 \times 6 = 18$ pounds per square foot, or 9 times the pressure required at 100 feet velocity. Figures 24, 25, and 26 show the relative pressures required on the area of the airway by change of velocity. Fig. 24 has 2 pounds per square foot, or a total pressure of 32 pounds for a velocity of 100 feet per minute.

Fig. 25 has $3 \times 2 = 6$ pounds per square foot to overcome three times the rubbing surface encountered per minute.

Fig. 26 has $3 \times 6 = 18$ pounds per square foot, being the 2 pounds raised to $3 \times 2 = 6$ pounds to overcome three times

FIG. 26.

18	18	18	18
18	18	18	18
18	18	18	18
18	18	18	18

Total 288 lbs.

the resisting surface, and this 6 again raised to $3 \times 6 = 18$ pounds to overcome the resistance caused by the triple momentum against three times the surface by three times the velocity. The velocity has been increased three times, and the pressure of 2 pounds has been increased $3 \times 3 = 9$ times. Nine is the square of three. Therefore the pressure *varies as the square of the velocity*.

The third, and perhaps the most important, principle involved in the friction of air in mines is the relation between the power expended and the result obtained, or the units of work or foot-pounds given out, and the velocity or quantity of air moved.

The velocity or quantity of air moved, all other things being equal, is in proportion to the *cube* root of the power expended,

and the power is, of course, in proportion to the *cube* of the velocity. It may seem strange that if a fan driven by an engine of 10 horse-power passes 20,000 cubic feet of air per minute, and it is desired to increase the quantity to 40,000 cubic feet, that it will be necessary for the engine power to be increased to $2 \times 2 = 4$ and $2 \times 4 = 8$ times 10, or 80 horse-power. If the quantity was increased to 60,000 feet, the engine would require a power of $3 \times 3 = 9$ and $3 \times 9 = 27$ times 10, or 270 horse-power. To properly comprehend this principle, it is important that the meaning of the terms "pressure" and "power" be clearly understood. "Pressure" is the force exerted on every square foot of the area of the airway, while "power" is this force multiplied by the amount of displacement it is causing, or the work it is accomplishing in a given time. As explained in "mechanics," work is the product of the force multiplied by the displacement it produces at its point of application. In this definition the force is supposed to be constant, and its point of application coinciding with the direction of the force. In expressing the work done by a force, the units of weight lifted through a unit of height, as in pounds lifted one foot, are called foot-pounds or work. The rate of working of a machine or the *power* it develops is expressed by the units of work done in a unit of time, as in foot-pounds per minute, or in conventional units called horse-powers. One horse-power is equivalent to 33,000 foot-pounds per minute, or 33,000 pounds lifted one foot high in one minute. Therefore a machine that can raise 12 tons through a height of 10 feet in 2 minutes is rather more than 4 horse-power, because 12 tons is 26,880 pounds, which multiplied by 10 feet is equal to 268,800 pounds lifted through one foot or foot-pounds in 2 minutes, and divided by 2 is equal to 134,400 pounds lifted through one foot in one minute, and this divided by 33,000 gives 4.07 horse-power.

To find the power exerted by a steam engine, the load or total pressure in pounds on the piston is multiplied by the number of feet the piston travels per minute with the load; the result represents the units of work or foot-pounds, and divided by 33,000 gives the number of horse-powers. On the same principle, when finding the horse-power exerted by a fan in moving

a current of air through a mine, the pressure in pounds per square foot, as represented by the difference of level in the two columns of the water-gauge, is multiplied by the number of cubic feet passing per minute; this result is the units of work or foot-pounds, which, divided by 33,000, gives the horse-power expended in moving the air current.

The water-gauge indicates the pressure on each square foot of area of the airway; therefore, the reason why the product of these two factors is the foot-pounds requires explanation. The quantity of air passing is found by multiplying the velocity in feet per minute by the area and again by the pressure to find the foot-pounds. There are only the three factors, the pressure, the area and the velocity, and changing the order of multiplication does not affect the result. Therefore, if we multiply the pressure by the area, which will be the total pressure, and their product by the velocity, the result will be the foot-pounds as before. In comparing the two cases, the area of piston in the engine would represent the area of airway under the fan, and the length in feet the piston travels in a minute represents the velocity of the air; the only difference is that with the fan we take the distance or velocity, and multiplying it by the area call it the quantity, while with the engine we take the three factors in succession. This may be further illustrated by an example.

An airway 2 feet wide and 2 feet high is passing 4000 cubic feet of air per minute with a pressure of 5 pounds per square foot; then the units of work found by the rule are pressure $5 \times$ quantity $4000 = 20{,}000$ units of work or foot-pounds. The area is $2 \times 2 = 4$ square feet, the total pressure is $5 \times 4 = 20$ pounds, and the velocity in feet per minute is $4000 \div 4 = 1000$ feet. Then the area $4 \times$ pressure $5 \times$ velocity $1000 = 20{,}000$ pounds per minute, or the same result. The total pressure represents the weight to be lifted, and the velocity or length of air column passing per minute the height it is lifted through, and their product is converted into horse-powers by being divided by 33,000. Hence, a fan passing 33,000 cubic feet per minute with one inch of water-gauge is expending 5.2 horse-power in moving the air current, and if for this work an engine of 10 horse-power is required, then we know that nearly 50 per cent.

of the power expended is absorbed in friction of the machinery and loss from imperfections in the fan. This is what is meant by *power* in ventilation, and that the power required varies as the cube of the velocity or quantity passing, and that the quantity passing varies as the *cube root* of the power expended, other things being equal, is demonstrated by the following example:

In an airway $2 \times 2 = 4$ square feet area passing 5000 cubic feet per minute with two pounds of pressure per square foot, the work expended will be

$$2 \times 5000 = 10{,}000 \text{ foot-pounds.}$$

To pass 15,000 cubic feet per minute, or three times the quantity, will require three times the velocity. It has been shown that three times the velocity requires nine times the pressure, which in this case would be 9 times 2 pounds, or 18 pounds per foot. Now, as the power is the pressure into the quantity, this $3 \times 3 = 9$ times the pressure multiplied by three times the velocity or quantity is $3 \times 9 = 27$, or the cube for 3 times the velocity or quantity. The work expended will be

$$18 \times 15{,}000 = 270{,}000 \text{ foot-pounds.}$$

$3 \times 3 = 9$, and $3 \times 9 = 27$,

and

$$27 \times 10{,}000 = 270{,}000 \text{ foot-pounds.}$$

270,000 is 27 times 10,000 units.

Therefore, three times the velocity or quantity of air requires 27, or the cube of 3, times the power, to move it; and if the power has to be increased as the cube of the velocity, then the velocity or quantity obtained by different powers will vary as their cube roots. Where a furnace is used for ventilating it will require eight times the coal to double the quantity of air in circulation, 27 times the quantity to triple it, 64 times the quantity to quadruple it, and so on in that proportion, and where a fan is used the coal necessary to raise steam will increase in the same ratio, as the same law controls both. The principle at first sight may seem complicated, but it is only the simple law of mechanics by which units of force, time, and space are measured into foot-pounds, and called "power."

This principle controls the resistance from friction, but

another principle involved is the force necessary to produce the velocity. The pressure required to cause the speed is equal to the weight of a column of air of such height that a weight falling from such height would acquire a velocity equal to the current. This has to be added to the pressure necessary to overcome the friction. The pressure necessary to generate velocity also varies as the square, but as this force is only a small fraction of the total force required to move air through mines, it is generally omitted in calculations. If the average velocity was 1000 feet per minute, the pressure necessary to generate such a speed under ordinary conditions would be

$$\frac{1000}{60} = 16.6 \text{ feet per second,}$$

and

$$\frac{16.6^2}{64.4} = 4.27$$

feet of air column. Assuming air to be 833 times lighter than water, this would be a water-gauge of

$$\frac{4.27 \times 12}{833} = 0.061$$

of an inch of water-gauge, a pressure so small when compared with the pressure of 1 to 2 inches found in mines that it may be omitted. This factor becomes less and less in proportion to the whole pressure as the mine extends.

The amount of resistance caused by sudden changes in the direction of the current is another source of resistance, but the amount is so small and the principles involved so complicated, that I have never seen them introduced as factors in problems in mine ventilation.

Ques. 10.—Give the formula for finding the pressure when the dimensions of the airway and volume of air passing per minute are known, and explain its meaning by an example.

Ans —The formula to find the pressure as required by the above question is

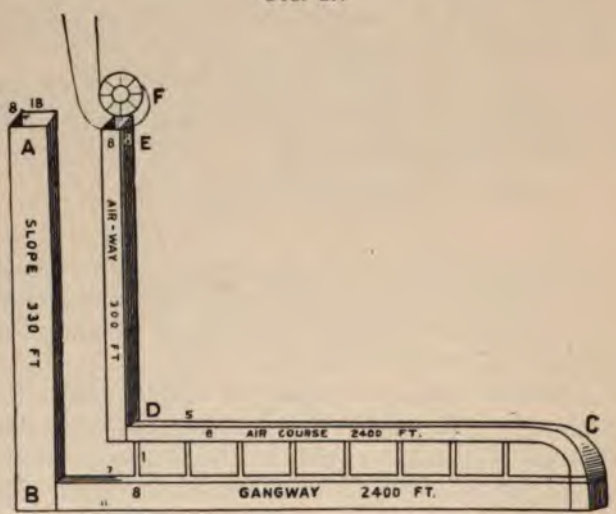
$$p = \frac{ksv^2}{a}.$$

Put into words this means, that if we take the velocity of the

air per minute and square it, multiply the square by the number of square feet of surface exposed to the moving air, and this again by the coefficient of friction, the result will be the total pressure on the airway putting the current in motion, and dividing this total pressure by (a) the area will, of course, give the pressure per foot. In order to illustrate the use of the formula take the following example:—

Let *AB* (Fig. 27) be a slope 18 feet wide, 8 feet high, and 110 yards long; let *BC* be a gangway 11 feet wide on bottom, 7 feet wide on top, 8 feet high, and 800 yards long; let *CD* be an airway 6 feet wide, 5 feet high, and 800 yards long, and *DE* a fan-return airway 8 feet wide, 8 feet high, and 100 yards long.

FIG. 27.



The volume of air as measured is 30,000 cubic feet per minute entering the fan, and on the supposition that there is no leakage in the slope or fanway the full volume is passing down the slope and up the fanway. As the full volume moves along the gangway, suppose it loses air at every cross-hole as it passes, and that when it reaches the inside end at *C* to return through the airway there are but 5000 cubic feet per minute. Hence but 5000 out of the original volume of 30,000 cubic feet reach the

last cross-hole, and 25,000 cubic feet are lost by leakage from the gangway to the return. The example could have been worked more easily on the supposition that the whole volume passed the whole length of the airways; but this is never the case in practice, and the problem as stated is as near an approach to what is found in our collieries as its nature will permit.

These are the terms of the example. Let us then ask the question, "*What is the water-gauge at the fan*" under these conditions, all other things being equal? To find the water-gauge the pressure per square foot must be found, and to find the pressure per foot the total pressure for the whole mine must be found. As the slope, gangway, airway and fanway are all of different sizes and lengths, the pressure putting 30,000 feet of air in motion will differ in each, and, therefore, the pressure per square foot for each must be found separately, the sums of which will be the whole pressure. As 5.2 pounds are equal to one inch of water-gauge, if the pressure thus found is divided by 5.2, it will give the water-gauge in inches or fractions of an inch, which is the amount required. Commencing with the fanway we proceed to find s, the rubbing surface; o, the perimeter, is

$$8 + 8 + 8 + 8 = 32 \text{ feet},$$

the sum of the four sides, which multiplied by the length is the rubbing surface:

l the length $= 100$ yards $= 300$ feet.

s the rubbing surface $= 32 \times 300 = 9600$,

the number of square feet of rubbing surface in the fanway.

The coefficient of friction generally used is

$$0.0000000217 \text{ lb.}$$

the part of a pound of pressure required to move the air over each square foot of surface at a velocity of 1 foot per minute; this multiplied by 9600, the number of square feet of rubbing surface, gives the whole pressure required to move a volume of air through the fanway at a speed of one foot per minute, which is a volume of 64 cubic feet per minute; as there are 64 cubic feet in every foot in length of the fanway.

This total pressure is skv^2, or

$$9600 \times 0.0000000217 \times 1^2 = 0.00020832$$

of a pound for a velocity of one foot per minute. The velocity due to 30,000 cubic feet per minute in the fanway is

$$\frac{30{,}000}{64} = 468 = v.$$

It has been explained that the pressure necessary to move air varies, not as the velocity, but as its square; therefore, having found the total pressure for a velocity of 1 foot per minute, the total pressure for a velocity of 468 feet per minute is found as follows:

$$1^2 : 468^2 :: 0.00020832 : 45.62 \text{ lbs.}$$

Putting this into words, it means that as the square of 1 foot per minute is to the square of 468 feet per minute, so are the pressures required for each. Now because 1^2 cannot change the result, it is omitted in the equation, and because the total pressure for a velocity of 1 foot per minute is the product of the coefficient k and the surface s, their product into the square of the velocity of 468 feet gives the total pressure, 45.62 pounds, to move 30,000 cubic feet per minute in the fanway. The pressure per square foot, or p, is found by dividing the total pressure by the area, which is the given formula, $p = \frac{k\,s\,v^2}{a}$.

The total pressure for the slope is—

Length, $l = 110$ yards $= 330$ feet.
Area, $a = 18 \times 8 = 144$ square feet.
Perimeter, $o = 2 \times 8 + 2 \times 18 = 52$ feet.
Surface, $s = 52 \times 330 = 17{,}160$ square feet.
Velocity, $v^2 = \left(\frac{30{,}000}{144}\right)^2 = 43{,}401.$

Therefore, $ksv^2 = 0.0000000217 \times 17{,}160 \times 43{,}401 = 16.16$ pounds as the total pressure for the slope.

The gangway being 4 feet narrower at the top than at the bottom, the sides have 2 feet of spread; the length of the slant side is therefore equal to the square root of the sum of the squares of the height and spread, or

$$\sqrt{8^2 + 2^2} = 8.24 + \text{ feet.}$$

The perimeter is therefore as follows:—

$11 + 7 + (2 \times 8.24) = 34.48$ feet $= o$.

The area is $\frac{11+7}{2} \times 8 = 72$ square feet $= a$.

The length is 800 yards $= 2400$ feet $= l$.

The surface is $2400 \times 34.48 = 82{,}752$ square feet $= s$.

The velocity at the outer end is

$$\frac{30{,}000}{72} = 416.6 \text{ feet.}$$

The velocity at the inside end for 5000 feet per minute is

$$\frac{5000}{72} = 69.4 \text{ feet;}$$

the average velocity * is therefore

$$\frac{416.6+69.4}{2} = 243, \text{ and } 243^2 = 59{,}049; \text{ therefore}$$

$ksv^2 = 0.0000000217 \times 82{,}752 \times 59{,}049 = 106$ pounds as the total pressure for the gangway.

The airway dimensions are

Length, $l =$ 800 yards $= 2400$ feet.
Area, $a =$ $5 \times 6 = 30$ square feet.
Perimeter, $o =$ $2 \times 5 + 2 \times 6 = 22$ feet.
Surface, $s = 2400 \times 22 = 52{,}800$ square feet.

Velocity at outer end of airway for 30,000 cubic feet is

$$\frac{30{,}000}{30} = 1000 \text{ feet.}$$

Velocity at the inside end for 5000 cubic feet is

$$\frac{5000}{30} = 166 \text{ feet.}$$

Therefore the average velocity is—

$$\frac{1000+166}{2} = 583, \text{ and } 583^2 = 339{,}889, \text{ then}$$

$ksv^2 = 0.0000000217 \times 52{,}800 \times 339{,}889 = 389.43$ pounds as the total pressure for the airway.

* Squaring the average velocity is not strictly accurate, but is sufficiently so for rough calculations.

The sum of these four pressures per square foot is:—

Fanway	45.62 pounds	÷	64 =	.71
Slope	16.16 "	÷	144 =	.11
Gangway	106.00 "	÷	72 =	1.47
Airway	389.43 "	÷	30 =	12.98
Total				15.27 pounds.

This pressure divided by 5.2 gives the height of water column or water-gauge it will balance:—

$$\frac{15.27}{5.2} = 2.94 \text{ inches,}$$

or about 3 inches of water-gauge (W. G.), which is the answer to the question. It is instructive to note the difference of total pressure required by each of the sections by comparison of pressure and dimension.

Ques. 11.—Give the formula for finding the volume or quantity of air in motion when the dimensions of the passages through which the air circulates and the height of the water-gauge are given.

Ans.—The formula to find the volume is generally expressed by

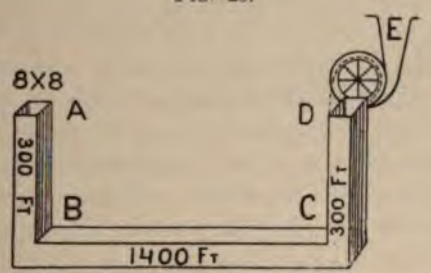
$$V^2 = \frac{Pa}{Ks}.$$

This means that if we take the amount of water-gauge in inches and change it into pounds per square foot (the quantity designated by P) and multiply it by a, the area of the air passage, the product will be the total amount of pressure on the passage, indicated by Pa. Then take s, the number of square feet of rubbing surface in the passage, and multiply it by K, the coefficient of friction, and divide the former product by the latter, the quotient will be the square of the velocity, the square root of which, of course, will be the velocity itself. Then the velocity and area multiplied together will give the volume or quantity of air put in motion under the given conditions, or

$$\text{Volume} = \sqrt{\frac{Pa}{Ks}} \times a.$$

In the first example we will deal with passages having a uniform section, and hence a uniform velocity, before solving more complicated examples in which air passages are of unequal sizes. Let *A B*, *B C*, *C D* (Fig. 28), be three air passages whose areas are 8×8 feet, and whose lengths are $300 + 1400 + 300 = 2000$ feet, thus making one continuous passage of $8 \times 8 = 64$ square feet area and 2000 feet in length. Let *E* be a fan placed at *D*,

Fig. 28.



exhausting or forcing air through these passages. Putting a water-gauge at the fan and finding it indicating $\frac{3}{4}$ inch, how much air is the fan putting into circulation under the above conditions?

As before stated, to find the volume we first find the velocity by the formula

$$\frac{Pa}{Ks} = V^2$$

which, since it is composed of four factors, we proceed to find their values as follows: Since P represents the pressure per square foot in pounds, we reduce the water-gauge, which is given at $\frac{3}{4}$ inch, to pounds, thus :—

$$\tfrac{3}{4} = .75 \times 5.2 \text{ lbs.} = 3.9$$

pounds per square foot.

The size of the passage being 8×8 feet, the ares (represented by a) is 64 square feet.

Consequently $3.9 \text{ lbs.} \times 64 = 249.6$ pounds, the total pressure on the area, or Pa.

Now find the value of Ks, so as to divide Pa by it. K, which

represents the coefficient of friction, is equal to 0.0000000217 of a pound pressure for each square foot of rubbing surface when the air is moving at a velocity of one foot per minute. S, the number of square feet of surface in the passage creating the friction or resistance that is to be overcome by the pressure, is found by multiplying the perimeter by the length.

The passages being 8×8, the perimeter, or the four sides, will be $8 + 8 + 8 + 8 = 32$ feet of surface for every foot of length; and $32 \times 2000 = 64{,}000$ square feet of surface in the whole passage.

Now, if there are 64,000 square feet of surface in the whole airway, and 0.0000000217 of a pound pressure is required for every square foot of surface to move the air one foot in one minute, it will require as many times 0.0000000217 of a pound pressure as there are square feet of surface in the airway to move a volume at that speed through it; or, to put it in different language, if every square foot of surface uses up 0.0000000217 of a pound pressure at a velocity of one foot per minute, we multiply 0.0000000217 by the number of square feet of surface, which, in this instance, we found was 64,000, and the product will be the total pressure required to move a column of air through the passage at a velocity of one foot per minute.

Hence 0.0000000217, or $K \times 64{,}000$, or s, $= 0.0013888$ of a pound or Ks, = the total coefficient pressure.

It has been demonstrated that a change of pressure produces a change of velocity, but that the amount of pressure required to make the change must be in proportion to the square of the velocity. That is, if 1 pound would produce a velocity of 1 foot, it would require 9 pounds to produce a velocity of 3 feet, 9 being the square of 3, and 1 being the square of 1.

For the same reason, if a pressure of 0.0013888 of a pound will move air through the above described airways at a velocity of one foot per minute, and the pressure is given by the conditions of the problem to be 249.6, the pressure will be proportional, not to the respective velocities, but to the square of such velocites. Hence the following equation :—

As the total pressure, Ks, due to a velocity of one foot per minute, or 0.0013888 pound, is to the total pressure, Pa, or

249.6 pounds, so is unity to the square of the velocity it will produce; therefore,

$$\text{Ks} : \text{Pa} :: V^2 : V^2$$
$$0.0013888 : 249.6 :: 1 : 179{,}579,$$

giving 179,579 as the square of the velocity this total pressure will produce, the square root of which is

$$\sqrt{179{,}579} = 423$$

the velocity itself.

This multiplied by the area gives the quantity or volume, which in this case is

$$423 \times 64 = 27{,}072$$

cubic feet per minute.

In the proportion the first term is Ks, the product of the coefficient and rubbing surface; the second term is Pa, the product of the pressure and area; the third term is unity, being the square of the velocity due to Ks, and the fourth is the square of the unknown velocity due to Pa.

Now, it is unnecessary to use 1 as a factor, since by using it as a multiplier or divisor the product or quotient is not changed. Therefore, we use the formula

$$\text{Pa} \div \text{Ks}, \text{ or } \frac{\text{Pa}}{\text{Ks}}$$

To make this still clearer, let us substitute simple quantities for the terms in the equation, and let Ks be two pounds, and Pa, the total given pressure, be 8 pounds. If the respective velocities produced by each vary in proportion to the square root of said pressures, then the pressures themselves must vary as the squares of the velocities. Now, we are supposing 2 pounds to produce a velocity of 1 foot per minute, then 8 pounds will produce a velocity whose square is, to the square of 1 as 8 is to 2, or

$$2 : 8 :: 1 : 4.$$

Remember, however, that the 1 and the 4 are not velocities, but their squares, and, as 1 is the square of 1, and 4 is the square of 2, then 1 and 2 are the velocities. This shows that the change of pressure from 2 lbs. to 8 lbs. has doubled the velocity, and that, as has been demonstrated, to double the velocity requires 4 times the pressure or the square of the velocity; and, as 8 is 4

times 2, it proves the proposition conclusively. If these substituted quantities in the equation prove the question for these quantities, they do it for the problem for the same reason.

We will now work a problem where the air passages are of different sizes and lengths, giving different velocities and resistances.

FIG. 29.

Let AB (Fig. 29) be a slope 14 $\times$ 8 feet area and 300 feet long; BC an airway 5 $\times$ 5 feet and 1400 feet long; CD a fanway 8 $\times$ 8 and 300 feet long. Let E be a fan placed at D the same as before. The water-gauge is $\frac{3}{4}$ inch.

Ques. 12.—How much air is the fan putting into circulation under the above conditions?

In the previous example the pressure produced a uniform velocity. In the present example it will produce a different velocity in each section. The whole length is the same as in the last example, but in the first three hundred feet it is much larger and the resistance is less, while the section from B to C is much smaller and its resistance is therefore increased, and only the section from C to D is the same as before.

Now, if we could find the relative amount of water-gauge, or pressure per foot, that is required to produce the current separately through each of the three airways, we could easily find the velocity or quantity, because, since we had the pressures as indicated by the said water-gauge, we could easily find the

velocities, and hence the quantities passing through each of the sections by the same formula,

$$V^2 = \frac{Pa}{Ks}$$

as we did before for the whole airway ; but as we do not know how much pressure is due to each section, we will have to assume some quantity passing, and then find the pressure required to pass said quantity by the formula

$$Pa = Ksv^2 \text{ or } P = \frac{Ksv^2}{a}$$

When the pressure for the assumed quantity is found, we can make proportions with the square root of the pressures, since the volumes vary in that ratio ; and

As the square root of the pressure of the assumed quantity is to the square root of the given pressure, so is the assumed quantity to the quantity that is passing.

Let the assumed quantity be 20,000 cubic feet per minute. Then we find the pressure per square foot for the assumed quantity in each of the several sections and add them together, and the sum will be the pressure for the assumed volume.

In the slope or section *AB*, multiply the perimeter by the length to find s, the surface : $8 + 8 + 14 + 14 \times 300 =$ 13,200 square feet : a, the area, is equal to

$$8 \times 14 = 112 \text{ square feet.}$$

V, the velocity of 20,000 cubic feet per minute, is

$$\frac{20,000}{112} = 178.5,$$

and the square of the velocity (V^2) is $178.5 \times 178.5 = 31,862.25$, then, $0.0000000217 \times 13,200 \times 31,862.25 = 9.127$ pounds, the total pressure, Pa, for section *AB*.

$$P = \frac{9.127}{112} = .08 \text{ of a pound}$$

per square foot.

For section *BC*, the total pressure for a volume of 20,000 cubic feet is found in the same manner.

$$a = 5 \times 5 = 25 \text{ square feet area.}$$
$$s = 4 \times 5 \times 1400 = 28{,}000 \text{ square feet surface.}$$
$$V^2 = \left(\frac{20{,}000}{25}\right)^2$$

$800 \times 800 = 640{,}000$ (the square of velocity). Hence, $0.0000000217 \times 28{,}000 \times 640{,}000 = 388.864$ pounds pressure, or Pa and

$$\frac{388.864}{25} = 15.55 = P.$$

For section *CD* the total pressure for 20,000 cubic feet is

$$8 \times 8 = 64 \text{ square feet} = a$$
$$4 \times 8 = 32 \times 300 = 9600 \text{ square feet} = s$$
$$\frac{20{,}000}{64} = 312.5 = V,$$

and $V^2 = 312.5 \times 312.5 = 97{,}656.25$, and $0.0000000217 \times 9600 \times 97{,}656.25 = 20.34$ pounds pressure, or Pa ; and

$$\frac{20.34}{64} = .32 = P.$$

The pressures per square foot of the three sections of the airway added together amount to :

$$.08 + 15.55 + .32 = 15.95 \text{ pounds}$$

per square foot, which would be the pressure at the fan if 20,000 cubic feet of air per minute were passing.

And

$$\frac{15.95}{5.2} = 3.07 \text{ inches water-gauge.}$$

Having found the pressure for the assumed 20,000 cubic feet per minute, we reduce, as before, the water-gauge given in inches to pounds, thus

$$\tfrac{3}{4} \text{ inch} = .75 \times 5.2 = 3.9 \text{ pounds.}$$

Since the volume varies as the square root of the pressure, we can make the following proportions, as

$$\sqrt{15.95} : \sqrt{3.9} :: 20{,}000 : 9{,}875$$

cubic feet, the volume which will pass through the three different sections when the water-gauge registers $\frac{3}{4}$ inch.

The principle is to work the problem with some assumed quantity, and then proportion from the resultant pressure.

Ques. 12.—Give your views and explain fully what is the best and cheapest plan to increase the volume and improve the ventilation of mines.

Ans.—The best and cheapest plan for improvement depends on so many different things that it is necessary to know the conditions existing before an intelligent system of improvement can be established; and all we can do within the scope of a question like this is to show by example what changing certain of these conditions will effect.

It is necessary that we should understand both natural and furnace ventilation, and be able to calculate their power when the difference of temperature they produce is ascertained, so that we may understand the principle involved in the movement of air currents.

In a colliery where the ventilation is imperfect in distribution, we may improve it by close attention to details, and generally at no very great expense; but where the deficiency is in volume, then we must look in some other direction for improvement.

As nearly all our collieries are now ventilated by fans, the first direction in which we naturally look for improvement is in "greater speed of the fan." In places where a fan is running slow, and there is plenty of power to run it fast, this can be done; but as this is only an increase of power, the extra volume obtained is not so much as might be supposed.

For example, let us suppose one of our steep-angled collieries down three lifts, with the airways driven on the regulation plan of about 25 feet area; the fan airway down for a thousand feet, about 50 feet area; and the current dragged up and down small manways. Suppose an 18-foot fan, running at a speed of 70 revolutions per minute, with an inch of water-gauge, is moving 63,462 cubic feet per minute. This is a greater efficiency than some of the fans in our districts are showing. If the slope is on the Mammoth seam, and a tunnel is driven to the Seven-foot and Buck Mountain seams, and the gangways are in a long dis-

tance, then from leakage and other causes the ventilation at the face of the gangways and breasts may be very poor, even with sixty thousand feet as measured at the fan. Under these conditions, this would be an expenditure of power in moving the current alone of 10 horse-power; because one inch of water-gauge = 5.2 pounds pressure and

$$\frac{5.2 \times 63{,}462 \text{ cubic feet}}{33{,}000} = 10 \text{ horse-power.}$$

As our fan is not supposed to be adjusted in the discharge opening to exactly suit the resistance at that speed, it may be giving out about 33 per cent. of useful work from the whole power of the engine. This would make the engine 30 horse-power.

We will now suppose the fan speed increased until the water-gauge rises to two inches = to 10.4 pounds of pressure. Now as the volume increases as the square root of the pressure the volume will be as follows:

$$\sqrt{5.2} : \sqrt{10.4} :: 63{,}462 : 89{,}626,$$

a change from 63,462 to 89,626 cubic feet per minute, or $\frac{1}{2}$ more air—the first inch of water-gauge showing twice as much air as the second inch, as it has only added 26,164 feet, or $\frac{1}{2}$ of the first, or $\frac{1}{3}$ of the whole result. Let us now see what power is required to do this:

$$\frac{2 \text{ in.} \times 5.2 \times 89{,}626 \text{ cubic feet}}{33{,}000} = 28.24 \text{ H. P.}$$

Supposing the same percentage of work given out by the fan —33 per cent.—then the engine would now require to be

$$28.24 \times 3 = 84.72 \text{ H. P.}$$

or nearly 85 horse-power, showing that a change from a 30 to an 85 horse-power engine will only give 20,000 feet more air under common conditions, and as an 85 horse-power engine is not common here, we at once see that increasing the power is a very limited and expensive plan of increasing ventilation. Therefore in a mine where the resistance is great and a small volume is obtained with a moderate-sized fan, to make any great improvement in ventilation by increasing the speed of the fan

would result in a great waste of power and require very powerful engines. Twenty thousand cubic feet extra divided up among, say, six different gangways and their breasts would be a very small improvement for so much more power. This shows that in a colliery where the airways are small in area and large in extent, where the water-gauge is very high for a small volume, of about 20,000 feet, that to put on a large fan and engine would not do so much good as the extra expense would warrant, and is not the proper direction in which to look for much improvement. Changing the construction of a badly arranged fan so as to get a little more percentage out of the engine will result much the same way; still we should try to get as much as possible out of the power expended, because a properly adjusted fan is as cheap as one which is not, and power wasted is money spent in vain.

This brings us to the next plan, namely, increasing the area. In large flat seams, airways of large section can be driven and large volumes of air at low velocity will insure good ventilation if the distribution is properly attended to. In locations where the Mammoth seam is on an angle of less than 40°, and strong hard coal, large airways can very easily be driven, either as a double gangway or as a large airway over the chutes; or by short lifts ventilated in pairs, one of each pair forming an intake and the other a return. As all, or nearly all, our collieries have worked the coal downwards in lifts from where it crops out on the mountain, with the hoisting and pumping openings placed as centrally in the territory to be mined as convenient, if an opening had been preserved from the surface down through each lift, at each end against the solid boundary pillar, for a discharge or intake, we would have in this district the simplest and easiest ventilated collieries possible. In locations where the coal measures are flat and at great depths, where the coal is only reached by shafts, the best arranged ventilation by large airways and judicious splitting can generally be attained. But the large volumes due to these causes are counteracted by the shaft resistances, which are increased as the square of the volume, because any change inside which would double the volume would increase the shaft resistance four-fold. In our district

the slopes and gangways are so large that they offer very little resistance, but the power is spent in small "dogholes," as they are very properly called, and small pillar headings. If it were not for the leakage of the current through batteries and brattices on its way inward, there would be very small volumes indeed. If the whole current had to pass the whole distance, with our small water-gauge pressure, we would have very small quantities of air in circulation. This leakage does for us what splitting does for the English, but in many places if we would lay out the ground in proper shape we could get the advantage of unequal splitting, at least to some extent.

Where the seams are thin on steep angles, it is impossible to drive either double gangways or airways over the chutes. If an opening can be driven up to the lift above, then the ventilation can be kept in average condition; but our lifts are generally too long. If a horizontal opening or gangway was driven every 200 feet, then the one could be very easily connected with the other, and if the first lift had been opened to the surface at short distances, the other lifts could be connected with the same openings and insure good results by making the airway short in extent. Where the coal is hard and the chain pillar below the water-level is preserved to keep up the surface water, air connections can be put through high enough above the drain ditch so that the water may be prevented from going down into the lifts below. Where the coal is soft and the pillars run out, then the water will all go down under any system of ventilation. For the purpose of showing by actual quantities the difference between large and small air passages in producing large volumes of air in mines, when their lengths are equal and the power applied remains constant, let us take the following examples:—

EXAMPLE 1.

Let A, B, and C be three airways, viz.:
A is $5' \times 5'$ and 1000 yards in length.
B " $7' \times 9'$ " " " " "
C " $10' \times 10'$ " " " " "

For illustration we will suppose that 10,000 cubic feet are passing through A without any leakage. With Atkinson's co-

efficient of friction, this would require a power equal to 83,328 units of work, or nearly 2¼ horse-power, to overcome the friction alone, without taking into account the power necessary to create a velocity of 400 feet per minute; because

$l = 1000$ yards $= 3000$ feet.
$a = 5' \times 5' = 25$ square feet.
$o = 5' \times 4' = 20$ feet.
$s = 20 \times 3000 = 60{,}000$ square feet.
$q = 10{,}000$ cubic feet per minute.
$v = \frac{10{,}000}{25} = 400$; and $400^3 = 64{,}000$.
$k = 0.0000000217$ lb.

then $\underset{s}{60{,}000} \times \underset{k}{0.0000000217} \times \underset{v^3}{64{,}000{,}000} = 83{,}328$, which divided by $33{,}000 = 2\frac{1}{2}$ HP. nearly.

Let us now see what this same power will put through the airway B, the dimensions of which are as follows:

$l = 1000$ yds. $= 3000$ feet.
$o = 2 \times 7 + 2 \times 9 = 32$ feet.
$s = 32 \times 3000 = 96{,}000$ square feet.
$k = 0.0000000217$ lb. per foot, and because the velocity is found by the formula

$$v = \sqrt[3]{\frac{u}{k\,s}}$$

then we have the following result from B:—

$$\sqrt[3]{\frac{83{,}328}{.0000000217 \times 96{,}000}} = 341 \text{ feet velocity.}$$

and $341 \times 63 = 21{,}483$ feet per minute for the airway B.

For the airway C the volume will be as follows:—

$l = 1000$ yards $= 3000$ feet.
$o = 4 \times 10 = 40$ feet.
$s = 40 \times 3000 = 120{,}000$ square feet.
$k = 0.0000000217$ lb.

Then by the same formula we used for B

$$\sqrt[3]{\frac{83{,}328}{.0000000217 \times 120{,}000}} = 318 \text{ feet velocity.}$$

v a
and $318 \times 100 = 31{,}800$ feet per minute.

Then we have

	Area.	Volume.
For A,	25 sq. ft.	10,000 feet per minute.
" B,	63 "	21,483 " "
" C,	100 "	31,800 " "

with the same power or units of work expended on each. If the pressure was constant, the difference in volume would be still greater.

These three airways show that the difference is great between a small and large airway for equal powers, but still the difference is not so much as some might expect from the great difference in their sizes, because if we enlarge an airway to double the area we lower the speed for the same volume, half of which would give one-fourth the resistance, provided the rubbing surface was not increased; but widening the airway must increase the resisting surface, which acts in the opposite direction and counteracts to a great extent the advantage of the decreased velocity. This shows that to increase the area is much better than to increase the power or the pressure. In collieries where the seams are thin and the dip steep, increasing the area is not always practicable.

In some text-books we see examples in figures showing the result of reducing the surface (s), and problems are given to show that to reduce the rubbing surface is to increase the volume in inverse proportion to the square root of the surface; but as I never could see any way to reduce the rubbing surface in mines, we will pass it over as a problem useful only for instruction in principle, but of course worth learning.

This brings us to the last and best plan for increasing the volume where it can be effected, namely:

SPLITTING THE CURRENT.

In our text books splitting is treated as equal and unequal, and examples are given by which we can learn its effects. These examples are worked by formulæ which require some study to understand. Equal splitting is on the assumption that the original current is cut up into equal divisions, all divisions to leave the main current and return into it at the same points. If the resistance of the intake and return, where the whole current enters before being split and leaves after the splits unite, are not taken into account, then the calculations are very simple. As this cannot be exactly done in practice, and because any increase gained by splitting inside causes a larger volume to be passed through the intake and return, this increased resistance causes the result to be a quantity less than the quantity due to the effect of splitting inside the mine.

The following simple example will show the effects of both equal and unequal splitting, and will serve to illustrate the principle involved, so that we can better appreciate the effects of splitting as found in our mines.

Let AB denote an airway under the following conditions:—

Area $= 5' \times 5' = 25$ square feet $= a$.
Length 3000 feet $= l$.
Passing 10,000 cubic feet per minute $= q$.

The pressure due to quantity is found by

$$P = \frac{k\ s\ v^2}{a}$$

The perimeter is $4 \times 5 = 20$ feet, which multiplied by the length is

$$20 \times 3000 = 60{,}000 \text{ square feet} = s.$$

The velocity is $\frac{10{,}000}{25} = 400$ lineal feet per minute.

The pressure is $\frac{.0000000217 \times 60{,}000 \times 400^2}{25} = 8.33$ pounds per square foot, which is equal to a power of

$$8.33 \times 10{,}000 = 83{,}300 \text{ units of work} = u.$$

The pressure being equal to $\frac{8.33}{5.2} = 1.6$ inches of water-gauge,

and the units of work to $\frac{83.300}{33,000} = 2.5$ horse-power.

To show the effect of equal splitting, let us assume the ventilating power applied at the middle and acting on both ends.

This changes the conditions to two airways of 1500 feet each.

Suppose the airway cut in the middle and the halves laid side by side, they may be considered as one airway of half the length, but double the area (a). The total rubbing surface is not changed, and the total pressure is spread over double the area, and, therefore, only $\frac{1}{2}$ as much per square foot. But the half pressure on each airway having only the friction of half the rubbing surface, which is the resistance, will maintain the same velocity and quantity in each as the double pressure caused in the whole length, thus passing double the total quantity. Again, because the pressure into the quantity is the unit, or power, half the pressure into double the quantity, or

$$\frac{8.33}{2} \times 20,000 = 83,300,$$

proves that the amount is constant.

Assuming the two halves as one airway, and finding the quantity by the formula

$$q = \sqrt[3]{\frac{u}{k\ s}} \times a,$$

gives the same result, because

$$\sqrt[3]{\frac{83,300}{.0000000217 \times 60,000}} \times 50 = 19,950$$

cubic feet per minute nearly, which proves the question, the slight difference being due to not carrying out the cube root to over three places in the decimal fraction.

If we divide the power between the two halves and work out the quantity by the same formula, we get the same result of 10,000 cubic feet in each.

The reason for this is much the same as if we had a rope in a long tube, and pulled it through at the end. If it required 50

pounds of force to pull 3 feet out every minute, by cutting a hole in the middle, and pulling the double rope, the same weight of rope would only need to be moved, but the same force would draw 6 instead of 3 feet per minute.

This shows the great difference in volume obtained by equal splitting, and proves that if it were not for the resistance in the return after the splits rejoin being increased from the larger volume, and slightly counteracting the effect of the splitting, we would obtain as much more air as we made splits. From causes which are present in all mines this theoretical result cannot be realized in practice. Still it comes sufficiently near it to teach us actually what is meant by splitting.

If the fan speed was increased after splitting until the pressure was the same as before, the difference would be still greater, because airways of unequal area, but equal rubbing surface, under equal pressure will pass quantities in proportion to $\sqrt{a} \times a$.

Hence, in this example, the result under constant pressure would be $\sqrt{25} \times 25 : \sqrt{50} \times 50 :: 10{,}000 : 28{,}280$ nearly, showing that with two equal splits under constant pressure the volume is increased nearly 3 times, and with constant power it is doubled.

Taking into account the power or pressure necessary to create the velocity independent of the resistance from friction, would introduce more factors and complicate the calculations without serving the purpose of illustration any better.

To show the effect of *unequal splitting*, we will assume the ventilating power applied at a point 1000 feet from the end. This is similar to splitting the air of a mine having double the extent of workings on one side than on the other, and may be taken as a simple example of unequal splitting.

This changing the position of the ventilating power changes the quantities in each by increasing the one and reducing the other.

The power, the area, and extent of rubbing surface are the same. But the resistance is proportional to the square of the velocities; and as the velocities in this problem are unknown, both the resistance and total quantity of air are unknown. But

both splits are from the same ventilator and are therefore under equal pressures. Now airways of equal areas and perimeters but unequal lengths under equal pressures pass quantities of air inversely as the square roots of their lengths. Therefore in this problem where the lengths are as 2 to 1, the quantity of air passing in the long split is to the quantity passing in the short split as $\sqrt{1} = 1$ is to the $\sqrt{2} = 1.41$. Or the total quantity of air passing is to the quantity passing in the short split as $1 + 1.41 = 2.41$ is to 1.41.

The simplest way to work this problem is to assume some total quantity of air passing, say 20,000 cubic feet per minute, then calculate the amount of work required by that quantity and from that amount of work calculate the quantity which the given amount of work would yield. Then

$$2.41 : 1.41 = 20{,}000 : 11{,}701$$

is the amount of air traveling in the short split, and

$$2.41 : 1 = 20{,}000 : 8299$$

is the quantity in the long split.

The velocity in the short split is

$$\frac{11{,}701}{25} = 468 \text{ feet per minute,}$$

and in the long split is =

$$\frac{8{,}299}{25} = 332 \text{ feet.}$$

The pressure required to pass the air in the short split is =

$$\frac{.0000000217 \times 20{,}000 \times 468^2}{25} = 3.82$$

and in the long split is

$$\frac{.0000000217 \times 40{,}000 \times 332^2}{25} = 3.82$$

pounds per square foot. The two pressures being equal as they should be.

The foot pounds of work per minute required to deliver 20,000 cubic feet of air is

$$3.82 \times 20{,}000 = 76{,}400.$$

But we want to know how much air 83,300 foot pounds will pass. As the quantity of air varies as the cube root of the work when all other things are equal, the actual quantities passing are found as follows:

$$\sqrt[3]{76{,}400} : \sqrt[3]{83{,}300} = 20{,}000 : 20{,}600.$$

total cubic feet of air delivered.

$$\sqrt[3]{76{,}400} : \sqrt[3]{83{,}300} = 11{,}701 : 12{,}050$$

cubic feet in short split.

$$\sqrt[3]{76{,}400} : \sqrt[3]{83{,}300} = 8{,}299 : 8{,}550$$

cubic feet passing in long split.

This change passes the larger quantity through the shorter side, where the least should be required, and would be imperfect distribution and poor management.

This brings us to another feature connected with splitting which will always present itself, especially in gaseous mines, *namely*, placing *regulators* in the short airways with a view to driving more air through the long ones.

To obstruct a short airway until its resistance is equal to a longer one, is practically the same as extending it to the same length, and for the purpose of calculation we assume such to be the case. This changes the condition from two airways of 1000 and 2000 feet respectively to two of 2000 feet each.

This change has increased the resistance as 4 is to 3. To find how much it has increased the pressure, and reduced the total quantity, so as to maintain the constant power, let us assume the two airways as one of 50 feet area and 80,000 feet of rubbing surface under a constant power of 83,300 units. By $\sqrt[3]{\frac{u}{k\ s}} \times a = q$, we find the quantity would be

$$\sqrt[3]{\frac{83{,}300}{.0000000217 \times 80{,}000}} \times 50 = 18{,}150$$

cubic feet per minute, with the regulator and constant power.

With this expenditure of 2½ horse-power nearly, let us compare the results:—

Before splitting,	10,000	cubic	feet	per	minute.
With equal splitting,	19,950	"	"	"	"
With unequal splitting,	20,600	"	"	"	"
In short airway,	12,050	"	"	"	"
In long airway,	8,550	"	"	"	"
With the regulator,	18,150	"	"	"	"
Quantity in each,	9,075	"	"	"	"

Showing that the regulator has only increased the quantity in the long airway 9075 — 8550 = 525 feet, and reduced the quantity in the short airway 12,050 — 9075 = 2975, a change of very little practical value.

Examination of these simple examples and resultant figures conveys some useful practical lessons. We can easily apply the ideas, and approximate the effect in practice, by studying the cause in theory, and thus learn the general principles involved in the ventilation of coal mines.

It further demonstrates that the whole subject of mine ventilation, stripped of its scientific wrappings, comes down to the simple statement: *powerful fans, wide airways and plenty of them!*

In showing, with given quantities, the effect of a regulator in this example, another question connected with the subject is naturally suggested, namely, determining the area or orifice in a regulator to produce a given result.

The term *regulator* is applied to a partition of boards or canvas placed across the return airway of the largest current, with an orifice to contract the current and increase its velocity at that point, until the resistance from the air having to rush through at such a high speed causes less to pass and forces more into the other airway.

The regulator does not increase the rubbing surface or friction, but causes resistance from velocity only, and because resistance is the equivalent of the pressure producing it, with given quantities we can find the area of the required orifice in square feet. Regulators increase the total resistance of the mine, and, therefore, their use is not commendable; but by

working problems of this kind we obtain clearer ideas on the subject of mine ventilation in general.

In our last example, the short airway passed 12,050 feet, and we wish to reduce that quantity to 9075 feet by making its resistance equal to that of the long airway. In that case the velocity through either airway is

$$\frac{9075}{25} = 363 \text{ feet linear.}$$

The pressure necessary to produce the velocity both with and without the regulator is found by

$$P = \frac{K\,s\,v^2}{a}.$$

Substituting known quantities, the pressure without the regulator is

$$\frac{0.0000000217 \times 20{,}000 \times 363^2}{25} = 2.29 \text{ pounds} = P.$$

The pressure with the regulator is the same as that required by the long airway or

$$\frac{0.0000000217 \times 40{,}000 \times 363^2}{25} = 4.57 \text{ pounds} = P.$$

and $4.57 - 2.29 = 2.28$ pounds, as the difference in pressure caused by the regulator, equal to a water-gauge or water column of $\frac{2.28}{5.2} = 0.44$ of an inch.

Assuming water at 833 times the weight of air, this water column of 0.44 inch is equal to an air column of

$$833 \times 0.44 = 366 \text{ inches, or}$$
$$366 \div 12 = 30.5 \text{ feet.}$$

This column falling freely through its own length attains a velocity of 8 times the square root of its length in feet per second,

$$\sqrt{30.5} \times 8 = 44.2 \text{ feet per second, or}$$

$44.2 \times 60 = 2650$ feet per minute.

This is the extra velocity imparted to the current by rushing through the orifice. The effect of the average velocity of 363 feet on the velocity through the regulator is so small that no

account need be taken of it. The areas are inversely proportional to the velocity in each, or 2650 : 363 :: 25 : 3.42 square feet as the size of the orifice. Practically, only about $\frac{2}{3}$ of the theoretical quantity will pass through an orifice; therefore it should be larger, or about

$$\frac{3.42 \times 3}{2} = 5.13 \text{ square feet, or nearly}$$

$\frac{1}{4}$ of the area of the airway.

The current authorities on mine ventilation do not appear to consider the subject of regulators of much importance to students, judging from the limited attention paid to it in the text-books. Erroneous ideas are common in regard to the effect of regulators. Therefore, we think the *regulator* should be included in the study of ventilation.

CHAPTER IV.

VENTILATION, CONTINUED.

Ques. 1.—In which of two openings, one 6×6 and the other 9×4, would the friction be greatest?

Ans.—The perimeter of the one is $4 \times 6 = 24$.

" " " other " $2 \times 9 + 2 \times 4 = 26$.

The friction in the opening 4×9 is to the friction in the opening 6×6 as 1.08 is to 1.

Ques. 2.—If you have the same reading of the anemometer in two openings, one 3 feet square and the other 9 feet square, how much more air is passing in the one than in the other?

Ans.— $3^2 = 9$, $9^2 = 81$,

$$\frac{81}{9} = 9$$

times more air in the large airway.

Ques. 3.—With furnace ventilation, how much more air will pass in a shaft 300 feet deep than in one 50 feet deep, other things being equal?

Ans.—The depths being as 1 is to 6, so is the pressure; therefore,

$$\sqrt{1} : \sqrt{6} :: 1 : 2.44,$$

or nearly $2\frac{1}{2}$ times more in the deepest shaft.

Ques. 4.—Which system of mine haulage requires the largest volume of air?

Ans.—Wire-rope haulage requires least, mule- or horse-power the next, and steam locomotive engines the most.

Ques. 5.—What is the area of a gangway where the collar is $6\frac{1}{2}$ feet between the notches, 2 feet spread, 8 feet high clear of rail?

Ans.— $6.5 + 10.5 = 17$,

$$\frac{17}{2} = 8\tfrac{1}{2},$$

$8.5 \times 8 = 68$ square feet.

Ques. 6.—What quantity of air is passing per minute in such a gangway when the anemometer registers 250 feet per minute, allowing 40 as the velocity for friction of the instrument.

Ans.—$250 + 40 = 290$, $290 \times 68 = 19{,}720$ feet per minute.

Ques. 7—What is the perimeter of such a gangway, and how many square feet of surface are contained in 700 yards of its length?

Ans.—To find the perimeter, we must first find the length of the legs. Let AB (Fig. 30) equal the leg, AC the height $= 8$ feet, BC the spread $= 2$ feet, then

$$AB^2 = BC^2 + AC^2 \text{ or } 8^2 + 2^2 = 68,$$

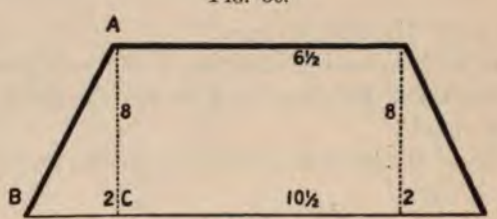
$$\sqrt{68} = 8.24 \text{ feet} = \text{length of legs.}$$

Both sides being equal, the perimeter is

$$6.5 + 10.5 + (2 \times 8.24) = 33.48$$

lineal feet $=$ the perimeter. The surface contained in 700 yards is $= 700 \times 3 = 2100$ feet in length, $2100 \times 33.48 = 70{,}308$ square feet.

FIG. 30.



Ques. 8.—What speed would the anemometer indicate in a circular opening 10 feet in diameter, when a volume of 10,000 cubic feet per minute is passing, allowing 40 feet velocity for friction of the instrument.

Ans.— $10^2 \times 0.7854 = 78.54.$

$$\frac{10{,}000}{78.54} = 127.32$$

$$127.32 - 40 = 87.32 \text{ feet,}$$

velocity indicated.

Ques. 9.—In a mine ventilated by natural means would you expect equal quantities of air at the inlet and outlet?

Ans.—With natural ventilation the openings are on different levels. In winter the air is expanded in the mine, and more bulk of air must leave it. In summer the air is condensed in the mine, and less bulk will pass out. With artificial ventilation the air is made lighter to its bulk and there will be more cubic feet at the outlet and a higher velocity through the same size of opening, neglecting gas produced in the mines. The quantity may be constant, but the volume is changed.

Ques. 10.—In a mine with two shafts, one 126 and the other 100 square feet area, which of the two would you make the upcast, if a furnace was used for ventilation?

Ans.—The resisting surface and average velocity through the mine would be the same in both directions, therefore the only difference would be in the shafts, because the air changes its bulk in the furnace, and only affects the shafts. The large shaft would allow more space for the expanded air, and by reducing the velocity keep down the resistance; but the expanded air has less weight to its bulk, the only difference is that the expanded bulk increases the speed as the bulk; and the resistance as the square of the speed; while the resistance from density only varies as the weight. According to this, if a given volume of air passes through a furnace and becomes double in bulk, it will be only one-half the weight for the same volume. Twice the bulk will give twice the velocity, and twice the velocity gives four times the friction; therefore, the resistance would be four times more from speed alone; but the air being only one-half the weight would have only one-half the friction for the same volume. We have, therefore, four times the increase in one way and one-half decrease in another, leaving the actual increase of friction only double or as the bulk, therefore the largest shaft would be the best as an upcast if it did not absorb any more heat; but the larger the shaft the more cooling surface, which has more effect than a slight change in speed. I would, therefore, make the *dryest* shaft the upcast, all other conditions being equal.

Question 11.—If there are two airways in a mine, one 5×8 and the other 5×6, with equal volumes of air passing through each, if the airway 5×8 is 1000 feet long, how long is the other?

Ans.—The airways being separate and in the same mine, may be supposed to be subject to equal pressure.

In order to find the length we must find the amount of surface in the airways equivalent in resistance to the total pressure by the formula

$$S = \frac{Pa}{Kv^2};$$

but as P and K are equal in both, these factors can be cancelled, and the formula becomes

$$S = \frac{a}{v^2}.$$

The areas are 30 and 40, and the quantity is equal for both. A velocity of 1 in the large airway would be a velocity of 1.33 in the small one. The surface in the large airway is

$$2 \times 5 + 2 \times 8 \times 1000 = 26{,}000 \text{ square feet;}$$

therefore, we have the following proportion:

$$\frac{4}{1^2} : \frac{3}{1.33^2} :: 26{,}000 : 11{,}022 \text{ square feet}$$

as the surface in the small airway. This divided by the perimeter will give the length—

$$\frac{11{,}022}{22} = 501 \text{ feet,}$$

as the length of the small airway.

Ques. 12.—If a mine employs 500 persons inside, and the air is split in two equal currents, what size would the airways have to be to pass the least quantity of air required by law, the velocity being 4 feet per second, the colliery located in the bituminous region of Pennsylvania?

Ans.—The law requires 100 cubic feet per minute for each person.

$$\frac{100 \times 500}{2} = 25{,}000 \text{ cubic feet}$$

in each split.

$$\frac{25{,}000}{60 \times 4} = 104.16 \text{ square feet area.}$$

$$\sqrt{104.16} = 10.2.$$

The airways, if square, would be 10.2×10.2 feet.

Ques. 13.—Two airways have each an area of 64 square feet. and their lengths are each 3000 feet, one circular, the other square: is their any difference in their rubbing surfaces? If so, how much?

Ans.—Square airway rubbing surface—

$$4 \times 8 \times 3000 = 96{,}000.$$

Circular airway

$$\sqrt{\frac{64}{0.7854}} \times 3.1416 \times 3000 = 85{,}011.7$$

Difference of surface 10,988.3 feet.

Ques. 14.—If you have two airways equal in dimensions, one 900 and the other 1600 feet long, passing a total volume of 25,000 cubic feet per minute, how much passes along each, both being separate splits from the same point?

Ans.—The quantity passing is inversely as the square root of the length; because the area and perimeter are equal in both, their lengths are as 9 is to 16.

$$\sqrt{9} + \sqrt{16} = 3 + 4 = 7;$$

then the sum of their square roots is inversely to the square root of each as the total volume is to the volume in each respectively;

7 : 3 :: 25,000 : 10,714.7 quantity passing in long airway.
7 : 4 :: 25,000 : 14,285.3 " " short "

Total 25,000 as before.

Changing the pressure with a constant rubbing surface, or changing the rubbing surface with a constant pressure, has the same effect on the quantity passing; the quantity changing as the square root in both cases where the areas and perimeters are equal.

Ques. 15.—If two airways have the same length and area, the one square and the other circular, with a total quantity of 30,000 cubic feet passing, what is the quantity passing in each?

Ans.—In this question the perimeter and length not being given, we have only the relation between the circumference of a circle and the perimeter of a square of equal area from which to proportion the rubbing surface. The circumference of a circle

whose diameter is 1 has a rubbing surface of 3.1416. The side of a square which is equal to a circle is 0.8862, diameter of circle being 1; therefore, the perimeter is

$$4 \times 0.8862 = 3.5448;$$

these being proportional to the rubbing surfaces, their square roots are inversely proportional to the quantities passing:—

$$\sqrt{3.1416} = 1.7724$$
$$\sqrt{3.5448} = 1.8827$$

Sum 3.6551

Then 3.6551 : 1.7724 :: 30,000 : 14,547.7
3.6551 : 1.8827 :: 30,000 : 15,452.7

Total passing 30,000.4 as before.

Ques. 16.—With natural ventilation can you insure a regular air current? If so, state fully your reasons. With a difference in depth of 40 feet in two shafts (ventilated by natural means), and also a difference in temperature of 40° between the inside and outside of a mine, show which shaft would be the upcast, supposing the outside temperature was highest, and give reasons why.

Ans.—Natural ventilation depends on the difference of level between the openings of the mine and the difference of temperature between the air inside and outside. The difference of height does not change, but the difference of temperature does. When the air outside and that inside are equal there is no pressure to produce a current; therefore, natural ventilation will not insure a regular current. With a difference of 40 feet between the levels of the openings and a difference of 40°, the motive column would be equal to

$$40 \times \frac{40}{459} = 3.48 \text{ feet.}*$$

The weight of $3\frac{1}{2}$ cubic feet of air would not produce any perceptible effect in moving air through a mine, although, theoretically, the air will move into the mine by the higher

* The temperature of the upcast not being given it is considered zero. The result is only approximate.

shaft when the outside air is the hotter, because the air is condensed in the mine when it is the colder and flows out by the lower opening. When the inside temperature is higher this is reversed, the cold outside air flowing in at the lower opening. When both temperatures are equal, there is no ventilation.

Ques. 17.—If there are two airways from the bottom of a shaft the full height of a 6-foot seam and 8 feet wide, the one 300 and the other 1200 yards long, to cause the long airway to pass the same quantity as the short one, how much must be taken off the side to widen it?

Ans.—Two airways commencing at a shaft are separate splits, and therefore subject to equal *pressure*. Their lengths are as 1 is to 4, and also their rubbing surfaces. The relative amount of air passing in the two airways will be inversely as the square roots of the lengths, because all other things are equal. We must, therefore, increase the volume passing in the large entry in that ratio.

But when the width is changed in the large airway the quantities of air passing before and after the change will be in proportion to

$$\sqrt{\frac{a}{s}} \times a,$$

because only the pressure and length remain the same. Now if the area is doubled the rubbing surface is increased, but is not quite doubled. But we can assume from an approximation that when the area is doubled or even trebled, that the rubbing surface is doubled or trebled. Then the volume of air in the long entry before and after doubling would be as

$$\sqrt{\frac{1}{1}} \times 1 \text{ is to } \sqrt{\frac{2}{2}} \times 2$$

or as 1 is to 2. Let us therefore assume that the quantities before and after the change are directly in proportion to the areas. Then for the new area:

$$\sqrt{1} : \sqrt{4} :: 48 : 96,$$

the area to which the long airway should be increased. Its height is 6 feet, therefore its width would have to be $96 \div 6 =$

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16 feet. The long airway would require 8 feet taken off the side and be widened to 16 feet, or double. This would make the perimeter of the long airway $2 \times 16 + 2 \times 6 = 44$ feet, and $44 \times 3600 = 158,400 =$ rubbing surface. The perimeter of the short airway is $2 \times 6 + 2 \times 8 = 28$, and its length is 900, and $900 \times 28 = 25,200$ rubbing surface.

$$\frac{158,400}{25,200} = 6.2858,$$

therefore their surfaces are now to each other as 1 is to 6.2858. The areas are also as 1 is to 2.

Airways under equal pressure pass quantities in proportion to

$$\sqrt{\frac{a}{s}} \times a.$$

If we apply this formula to both, the results should be equal, as they are to pass equal quantities.

Long airway,

$$\sqrt{\frac{2}{6.2858}} \times 2 = 1.12.$$

Short airway,

$$\sqrt{\frac{1}{1}} \times 1 = 1.$$

This shows that the area has been made too large, as it would pass 12 per cent. more air. This is near enough for all purposes, but the difference results from considering the quantity of air to increase as the area.

Ques. 18.—If there is a difference of pressure equal to 3 pounds per square foot between a downcast and an upcast shaft with a volume of 24,000 cubic feet of air passing per minute, what would be the difference of pressure with a volume of 48,000 cubic feet passing, other things being equal?

Ans.—The volume varies as the velocity, and the pressure as its square. In this question the volume is doubled, and, of course, twice the velocity requires 4 times the pressure; therefore the 3 pounds per square foot require to be increased to

$$3 \times 4 = 12 \text{ pounds},$$

a difference of 9 pounds per square foot.

The power expended for 24,000 cubic feet is

$$\frac{24{,}000 \times 3}{33{,}000} = 2.18 \text{ HP.}$$

The power expended for 48,000 cubic feet is

$$\frac{48{,}000 \times 12}{33{,}000} = 17.45 \text{ HP.}$$

$$\frac{17.45}{2.18} = 8;$$

therefore, the double volume requires 8 times the power.

In this proportion the quantity of air is doubled; therefore, twice the velocity requires

$$2^2 \times 3 \quad = 12, \text{ the pressure,}$$

and

$$2^3 \times 2.18 = 17.44 \text{ horse-power,}$$

the power, which proves that the volume of air passing in a minute varies as the square root of the pressure and the cube root of the power.

Ques. 19.—If an expenditure of 5 horse-power moves a current of 30,000 cubic feet per minute through a mine, what power will be required to move a current of 90,000 cubic feet per minute?

Ans.—The volume varies as the cube root of the power; 90,000 is 3 times 30,000, therefore three times the velocity would be required.

$$3^3 = 27;\ 27 \times 5 = 135 \text{ horse-power,}$$

three times the volume requiring the power to be raised from 5 to 135 HP. This shows that it is impracticable to obtain three times or even twice the air at our collieries by merely increasing the speed of the fan, because they are not driven by engines of sufficient power.

Ques. 20.—With a difference of one-tenth ($\frac{1}{10}$) of an inch of mercury as indicated by the barometer between a downcast and an upcast shaft, what would be the pressure per foot, and what would be the motive column, the temperature of the downcast averaging 60°?

Ans.—A cubic inch of mercury weighs 0.4908, or nearly ½ pound.

$$\frac{0.4908 \times 144}{10} = 7.06752$$

pounds per square foot as the difference of pressure. Air at 60° under 30 inches of mercury column weighs 0.0766063 pound per cubic foot.

$$\frac{7.06752}{0.0766063} = 92 \text{ feet}$$

as the motive column.

Ques. 21.—What is the size of an airway passing 16,000 cubic feet per minute at a velocity of 400 feet per minute?

Ans.—

$$\frac{16{,}000}{400} = 40,$$

$$\sqrt{40} = 632 +.$$

The airway will require to be a fraction over 6 feet wide and 6 feet high.

Ques. 22.—If 2 pounds pressure per foot produce 15,000 cubic feet per minute, how much will the pressure have to be increased to produce 30,000 cubic feet per minute?

Ans.—The amount of pressure required for different volumes of air is proportional, not to the volumes, but to their squares. To increase the volume from 15,000 to 30,000 is to double it. The volumes will therefore be as 1 is to 2, and the pressures will have to be as 1^2 is to 2^2. The pressure was 2 pounds, which will have to be increased to

$$2 \times 2 = 4 \text{ times } 2, \text{ or } 8 \text{ pounds per foot.}$$

Ques. 23.—If 2 horse-power produce 10,000 cubic feet of air per minute, how many horse-power will be required to produce 30,000 cubic feet per minute?

Ans.—The power necessary for different volumes of air is proportional, not to the volumes, but to their cubes. To increase the volume from 10,000 to 30,000 is to treble it, or, the volumes are as 1 is to 3; the power would therefore require to be as 1^3 is to 3^3. The power is 2 horse, which would have to be increased to $3^3 = 27$ times 2, or 54 horse-power. This is a

startling difference of power between the production of 10,000 and 30,000 feet of air. 2 horse-power are equal to

$$2 \times 33{,}000 = 66{,}000 \text{ units of work,}$$

which divided by the quantity gives the pressure putting it in motion.

$$\frac{66{,}000}{10{,}000} = 6.6$$

pounds per square foot of pressure. To produce 3 times the air would require the square of 3, or 9 times the pressure, which would be $3^2 \times 6.6 = 59.4$ pounds per foot for 30,000 feet; and as the pressure into the quantity is the unit of work, this would be

$$\frac{59.4 \times 30{,}000}{33{,}000} = 54$$

horse-power as before. The above shows the reason of this immense difference in the relative power required to move different volumes of air through the working of a mine. It also shows that the quantity of power given for 10,000 feet is not within the limits of practice, as such conditions should always be stated. No ventilating power existing at any mine is able to increase the pressure from 6.6 pounds to 59.4 pounds per square foot. (This question was asked by a board of examiners.)

Ques. 24.—Where would an exhaust fan give the best results —at the top or bottom of a shaft? Would there be any difference in the results, and why?

Ans.—If a fan was placed in the bottom of a very deep shaft, it would be acting both as an exhaust and a forcing fan. If the colliery was new and the shaft deep, the resistance of the upcast through which the fan acted as a blower might be half the resistance of the workings of the mine through which the fan acted as an exhauster; the fan would therefore be $\frac{1}{3}$ a forcing and $\frac{2}{3}$ an exhausting fan.

As far as mere mechanical effect of the fan is concerned, we do not think there is much, if any, difference whether it blows into, or exhausts out of, a mine, or is placed in the mine to exhaust from one part and blow into another. We are acquainted with fans placed in all three positions, and the efficiency of the

fans in all three cases is practically the same when the different conditions surrounding each are duly considered. The question as to where a fan should be placed to suit the special conditions of a mine is a very different problem. The reason that fans are generally placed to exhaust from the mine is that it leaves an open avenue for the air to pass down the shaft or slope and and along the main roads to the locality where the coal is being mined, thus avoiding the use of doors on the main intake currents, the smoke and gas from the working places being carried off to the fan in return airways, where they will not pass through other working places or come in contact with lights. This argument is more theoretical than practical. If a colliery is supplied with from 75,000 to 100,000 cubic feet of air properly distributed, the return currents will generally be safe with open lights; but the main benefit of the exhaust fan is in keeping the main roads clear of smoke.

One grave objection in our climate is the formation of ice in the shafts and slopes in winter. In a wet shaft, 100,000 feet per minute of air below the freezing-point form a mass of ice from every dropper and cause much danger and expense. Fans blowing into a mine overcome the difficulty of ice in winter, but the main roads are often smoky. If the fan is blowing inward along the gangway, main doors are required to disconnect the hoisting shaft, and the opening of the doors for every trip breaks the current. Double doors cannot always be placed a sufficient distance apart to properly overcome this difficulty.

Where a fan has to exhaust up a confined compartment of a shaft, by placing the fan at the bottom and using the whole shaft for the upcast it reduces the resistance and increases the volume; but the traffic would have to pass through doors at the bottom, and the amount lost by opening of the doors might be half the gain from this arrangement, during working hours at least, although at night the fan would give the full benefit in clearing the mine. There are mines where the fan would give the best results by being placed inside; but there is one grave objection, especially in gaseous mines—when placed *in* the mine the fan is, like the furnace, *out of reach of those on the surface* when any serious accident occurs.

Fans can be, and sometimes are, placed on the surface to exhaust from a hoisting shaft by enclosing the landing inside of an air-tight house over the shaft, with the ropes passing through the roof, but the cars have to be passed through double doors or air-locks, the fan being connected with the shaft under the landing. With every car that passes in or out there is generally considerable leakage; but the principal objection is the inconvenience of having a working shaft hemmed inside of an air-tight house. With deep shafts we would prefer this plan to placing the fan at the bottom, where it is necessary to exhaust from a hoisting shaft.

In answer to this question, we think that the difference of result in fans placed at top or bottom of a shaft, depends on the conditions existing in the mine, and not on the fan as a machine imparting motion to the air.

Ques. 25.—With the same power in operation, how can you increase the amount of ventilation in a mine?

Ans.—By splitting up the current into several sections, by allowing air to leak where the leakage will supply air where it is needed, by inspecting the whole course the air has to travel, and whenever a high velocity is found trying to reduce it by removing obstructions or knocking out old stoppings, so as to allow the current to pass through two or three headings instead of one, and driving all air passages as large as is consistent with safety and economy.

Ques. 26.—How would you arrange your airways to give the best results as to the quantity and quality of the air, and the safety of the employés, in a mine generating large quantities of explosive gases?

Ans.—By conducting the air in separate currents to panels or groups of working places, and always so that the return air would ascend where ascensional distribution was practicable.

Ques. 27.—Suppose your airways to be 4900 feet in length and you get 12,000 feet of air per minute, would you get more or less air by increasing their lengths to 6400 feet? How much more or less?

Ans.—The air volume decreases as the length of the air passage increases, but not in the same proportion. If the airways

are made longer, without changing their shape or area, then to produce the same quantity, the pressure would have to increase as the length; but if the pressure remained constant, then the quantity would decrease inversely as the square root of the length, because the volume varies as the square root of the pressure, and the rubbing surface or length is equivalent to the pressure required to overcome it, as long as the speed is not changed by a change of area.

In this question the quantity would be reduced to

$$\sqrt{6400} : \sqrt{4900} :: 12{,}000 : x \quad \text{or}$$
$$80 : 70 :: 12{,}000 : 10{,}500$$

cubic feet by extension of the airways, or in direct proportion to the reciprocal of the square roots of their lengths.

Ques. 28.—If your fire-boss reported to you that the furnace had been neglected during the night, and that he had discovered large quantities of fire-damp in the workings, what would be your method of procedure in order to clear out the gas?

Ans.—Break open a connection from the upcast shaft behind the furnace to the return, by which the gas would reach the upcast without passing through the furnace, and open the door between the downcast and the furnace to admit fresh air direct to the fire before firing up and starting the current. If this was impracticable owing to the location of the furnace, then I would open a stopping just outside of the gas and cause the fresh air from the intake to rush through in volume and allow a small quantity of fire-damp to mix with it, thus working it off slowly by diluting it to a safe mixture before reaching the furnace, clearing out one chamber after another, opening and closing the necessary stoppings in succession. If there was any waste containing black-damp through which the return air could be passed, or from which damp could be drawn to mix with the returning gas, I would mix them by opening a stopping, and thus damp the gas to make it non-explosive, but would on no account attempt to move it in large volumes.

Ques. 29.—Of what material would you construct permanent stoppings or brattices to secure duration and economy? Give reasons in full.

Ans.—Permanent stoppings should not be constructed of timber. They may be cheaper at first, but they soon become leaky from vibration of shots, and soon require repairs and renewal. The great objection is that they are neither fireproof nor strong enough to resist the force of small local explosions, which are apt to occur in even well regulated collieries which give off gas. If hard slate or stone can be obtained from the roof or floor of the mine, I would build all permanent stoppings of such, by facing up a wall with the best shaped stones laid in mortar, using a few bricks for special work; then pack behind with stone and debris to give support. Where building material could not be obtained in the mine I would use brick for the walls and support them with packing of any loose débris at hand. Where the coal is always crushing and squeezing, as in some of the anthracite mines, some argue that brick would be crushed, and to carry such heavy material up steep angles would be very inconvenient. This is partly true, but any stoppings which would come under the head of permanent are generally within easy reach of the gangways. Walls built of broad pieces of strong slate laid in mortar stand more crushing than any other material we have seen tried. We know from experience that permanent stoppings should be built of stone or brick, or both, as a measure of safety and economy in mines where dangerous gases have to be dealt with, and in fact in all coal mines with which we are acquainted.

Ques. 30.—What are the principal points to be kept in view in hanging doors to assist the ventilation in mines? State how you could ventilate a mine without the excessive use of doors. Give your methods and reasons fully.

Ans.—The frames of mine doors are stood slightly off the perpendicular to make them close by gravity; when pushed open they swing back and close unless they are blocked. This is required by law, to prevent them from standing open if left so by careless persons neglecting to close them. The main doors should be duplicated, one being fastened back against the wall ready to be closed in case of accident to the other. The frame should be built in with brick and made strong. In the anthracite coal region the doors are 7 to 8 feet wide and about the same

in height. A ventilating pressure of $\frac{1}{2}$ to $\frac{3}{4}$ inch of water-gauge against such a door requires some force to open it, and when allowed to slam shut it soon strains the door, causing it to leak and to require frequent repairs. This cause of annoyance and expense is avoided by double doors placed so far apart that one is closed behind a trip of cars before the other in front is opened, and this the law requires.

A door 9 or 8 feet wide must be nicely hung so as to swing across a mine track and at the same time fit air-tight on its frame. In some places the gangway is always squeezing and breaking the timber, the bottom heaves up and the sides and top squeeze in. Gangs of men are constantly at work, taking out broken timber, lowering the bottom and putting in new timber. In such places gas is plenty and doors are required. The doors have to be frequently lifted off their hinges and a piece sawed off to make them close; the bottom heaving, crushes them out of place. In this case the doors are made in two leaves, which meet in the middle of the track, this making them much easier handled and repaired, and not so easily jammed from heaving.

The fewer doors in a mine the better, and although there will always be some doors required for proper distribution, the excessive use of them can be avoided, by splitting the current into separate districts, and carrying the return air currents over the intake currents by air bridges or overcasts.

Ques. 31.—With furnace ventilation, what effect will a wet upcast shaft have on the ventilation of a mine, and would difference of depth have any bearing on the amount of air in circulation?

Ans.—A wet shaft would absorb heat and lower the temperature, thus counteracting the effect of the furnace. If all other things are equal, the depth of the shaft is a measure of the pressure, and the quantity varies as the square root of the pressure, and, therefore, as the square root of the depth. To double the depth would increase the volume as

$$\sqrt{1} : \sqrt{2}, \text{ or as } 1 \text{ is to } 1.41$$

or less than $\frac{1}{2}$ more air.

Ques. 32.—With three airways 3 feet by 4 and one large air-

way 6 feet by 6, which will give the most air with the same power applied, and what would be the difference, and why?

Ans.—The sum of the areas of the three airways is

$$3 \times 4 \times 3 = 36 \text{ square feet.}$$

The sum of their perimeters is

$$[2 \times 3 + 2 \times 4] \times 3 = 42 \text{ feet.}$$

The area of the large airway is 36, and its perimeter is

$$4 \times 6 = 24 \text{ feet.}$$

The areas are equal; therefore the quantities passing will be in inverse proportion to the cube roots of the perimeters.

$$\sqrt[3]{42} = 3.4 \text{ and } \sqrt[3]{24} = 2.8,$$

and

$$3.4 : 2.8 :: 1 : 0.82 +;$$

therefore the quantity that would pass in the 6×6 would be to what would pass in the three small airways as

5 is to 4, nearly.

Ques. 33.—If in a heading you encounter a "blower" which gives off 3000 cubic feet of fire-damp, and the volume of air passing is 12,000 cubic feet per minute, what would be the result if such a mixture came in contact with open lights?

Ans.—If 12,000 feet of air were mixed with 3000 feet of gas, the proportion would be 1 in 5. A mixture of 1 in 6 will not explode, but will extinguish lights, and as 1 in 5 is still nearer pure gas, such a mixture will extinguish the light of the lamp, and soon extinguish the life of the man who carries it.

Ques. 34.—If while exploring old or abandoned workings you found yourself on the top of a large fall (say 35 feet above the level of the roof of the mine) and your light should be extinguished by coming in contact with a *gas*, what in your opinion would be the kind of gas encountered, and what means would you use to satisfy yourself as to the identity of the gas in question?

Ans.—Under the above conditions I would suspect the gas was fire-damp mixed with black-damp by diffusion. If it had the peculiar odor of burning wood, I might suspect fire in the vicinity. Such conditions would exist only in abandoned workings, where there was no ventilation. In order to satisfy myself

I would pass a current under the gas, and then examine with a safety lamp. The fresh air would carry off the lower stratum of the mixture containing the greatest percentage of black-damp and render the fire-damp left above explosive, when it would show itself in the safety-lamp. If it smelled of carbonic oxide, I would not re-enter the place till the gas was removed. Practical miners do not find themselves on top of falls in abandoned workings with open lights; therefore the conditions laid down in the above question are not likely to occur.

Ques. 35.—For gaseous mines what do you consider the best mode for producing ventilation? Give your reasons in full.

Ans.—I would consider the fan as the best mode of producing ventilation, because it is the most reliable; the fan can be made to move the largest volumes at least expense under every and all conditions. It removes the danger of gas coming in close proximity to a fire as with furnaces, is perfectly under control, and not in itself much affected by change of temperature or pressure. And practically the fan is the only ventilator worthy the name which has as yet been applied to mine ventilation. I would, therefore, prefer fan ventilation for all mines, gaseous or otherwise.

Ques. 36.—What is the area of an aircourse 6 feet at top, 8 feet at bottom, and $6\frac{1}{2}$ feet high?

Ans.—$\frac{6 + 8}{2} \times 6.5 = 45.5$ square feet area.

Ques. 37.—State the principles on which ventilation depends. State also the cause of the air's motion, and what are the great factors of resistance operating against ventilation.

Ans.—Ventilation is produced by difference of pressure, and gravity causes the air to move in the direction from the heavier to the lighter, the same as water and other fluids. Of course the force of gravity, or attraction, is the primary cause which produces motion in the air; and when that motion is guided by artificial means it is called a current, and its effect of changing the air by displacement in a mine is called ventilation. The force stored up in the air as velocity or motion is absorbed by rubbing against the surfaces of the passages through which it is moving, and is called friction, which is the same resistance that

is produced by any two bodies moving in contact; this is one of the greatest factors of resistance. Another factor is the inertia to be overcome in moving the air columns. In mines where the openings are on different levels and where mechanical ventilation is used, the natural ventilating force produced by difference of temperature sometimes acts adversely to the mechanical force and often becomes a powerful factor of resistance. If they both act in one direction, then the resultant velocity is proportional to the square root of the sum of both the pressures. The principal factors operating to retard ventilation are small airways and high speed of the current. Any means which enlarges the passages, either by widening them or breaking them up into sections or splits, so that a number will act at the same time and supply different districts, will remove some of the principal factors of resistance to ventilation.

Ques. 38.—A square airway, 56 feet area, is passing 15,000 cubic feet of air per minute. It is required to increase the volume 5000 feet per minute; how much must the airway be enlarged, the power to remain constant?

Ans.—The formula for work is total pressure multiplied by velocity, which is:

$$u = ksv^3.$$

If we were to increase the area until the required amount of air passed at the same velocity as the original amount passed; the powers would be the same if the rubbing surface had not been increased. Let us assume for a trial that the new velocity is equal to the old velocity, then the areas are proportional to the volumes, or

$$3 : 4 = 56 : 74.66$$

which gives approximately the new area.

The perimeters are:

First airway, $\sqrt{56} = 7.48$; $7.48 \times 4 = 29.92$.

Second airway, $\sqrt{74\ 66} = 8.64$; $8.64 \times 4 = 34.56$.

Now because the velocities are equal the powers are to each other as the rubbing surfaces or as

$$29.92 : 34.56 = 1 : 1.15.$$

This shows that the new entry requires 15 per cent. more power than the old one. An increase in the area, when the volume is the same, decreases the cube of the velocity much faster than it increases the rubbing surface. Hence for a new trial we can neglect the rubbing surface and say that the powers vary as the cube of the velocities. But the velocities are inversely proportional to the area when the quantity is the same; so we may say that the cube roots of the works are inversely proportional to the areas or

$$\sqrt[3]{1} : \sqrt[3]{1.15} = 74.66 : 78.2.$$

Therefore the area of the new airway will be about 78.2 square feet. This is very close to result when the question is solved by more complicated methods.

Ques. 39.—Supposing there are two airways, one 4 × 12 feet, the other 8 × 8 feet: how much more air will pass in the one than in the other, with the same pressure?

Ans.—As no quantity is given, and no pressure or length to find a quantity, all we can do is to show the relation between the quantities that will pass under equal pressures.

The rubbing surfaces being equal, they will pass quantities under equal pressure according to

$$\sqrt{a} \times a$$

$$\text{First airway, } \sqrt{48} \times 48 = 332.16.$$

$$\text{Second airway, } \sqrt{64} \times 64 = 512.$$

$$\frac{512}{332.16} = 1.54.$$

The quantity that will pass into the large one being $\frac{1}{2}$ more than in the small one.

Ques. 40.—If there are two airways, one 4 × 4 and the other 8 × 8, how much more air would be got in the larger one, allowing 5000 feet per minute for the small one, pressure and length being the same?

Ans.—The quantities will be according to

$$\sqrt{\frac{a}{s}} \times a$$

First airway,

$$\sqrt{\frac{16}{16}} \times 16 = 16.$$

Second airway,

$$\sqrt{\frac{64}{32}} \times 64 = 90.496.$$

16 : 90.496 :: 5000 : 28,280 feet per minute in the large airway.

Ques. 41.—Give dimensions and description of a furnace, shaft, and stack required to circulate a lawful amount of air in a mine having 2000 yards of airways of thirty-six square feet area, working 100 miners, together with the usual number of drivers and mules; and what would be the kind and size of a fan to produce the same amount of air current in such a mine?

Ans.—Allow 100 cubic feet per minute for each person employed. Therefore, in this case $100 \times 100 = 10{,}000$ feet would be the lawful quantity for the miners; this force should produce 400 tons of coal, and would require 8 mules and 4 drivers; assuming a mule equal to 6 men, $6 \times 8 = 48$ and 4 drivers = 52, or in round numbers the whole force would be equal to 150 men, and require $150 \times 100 = 15{,}000$ cubic feet as the volume demanded under the law. The airway of 36 feet would give a velocity of current equal to

$$\frac{15{,}000}{36} = 416.6$$

linear feet per minute. Assuming the airway to be square, it would be 6×6, and its perimeter would be $4 \times 6 = 24$ feet, and 2000 yards equal 6000 feet in length; therefore, its rubbing surface would be

$$24 \times 6000 = 144.000$$

square feet; assuming the usual coefficient of friction, this would require a pressure of

$$\frac{0.0000000217 \times 144{,}000 \times 416.6^2}{36} = 15.0646$$

pounds per square foot, which would be a water-gauge of

$$\frac{15.0646}{5.2} = 2.897 \text{ inches.}$$

This is on the assumption that the whole volume travels the full length, which in practice it does not, nor is it desirable that it should. We may safely assume that this theoretical water-gauge of nearly three inches will in practice be reduced by leakage of the air through brattices, etc., to say 1.5 inches, which is about the average pressure produced by fans at mines. This would be

$$1.5 \times 5.2 = 7.8 \text{ lbs. per square foot.}$$

Two furnaces 6 feet wide and 4 feet high would have an area of $4 \times 6 = 24 \times 2 = 48$ feet, and

$$\frac{15{,}000}{48} = 312.$$

Therefore, the air in passing over such furnaces would have a velocity of about 300 feet, and a higher speed would be inconvenient. If gas was to be handled, then a dumb drift would be necessary, which would lower the speed through the furnace according as it was regulated. Supposing the temperature was 50° at the surface, and the furnace raised it to 200°, by the time it reached the top it might be cooled to 100°. This would be an average in the upcast of 150°. Air at 50° weighs

$$\frac{1.32529 \times 30}{459 + 50^\circ} = 0.0781113 \text{ lb. per cubic foot,}$$

and air at 150° weighs

$$\frac{1.32529 \times 30}{459 + 150^\circ} = 0.0652852 \text{ lb. per cubic foot,}$$

and $0.0781113 - 0.0652852 = 0.0128261$ lb. as the difference in weight per cubic foot between the air in the down and upcast shafts. The pressure required being 7.8 pounds, it would take as many feet depth of shaft as 0.0128261 is contained in 7.8 pounds,

$$\frac{7.8}{0.0128261} = 608 \text{ feet.}$$

Therefore, 2 furnaces 6 feet wide and 4 feet high at the bottom of a shaft 600 feet deep and 6×6 would produce the required ventilation under ordinary conditions.* The application of a

*The resistance of the shaft itself is here omitted because it is small compared with the resistance of the rest of the mine.

stack at the top of the shaft would add so little to the current that it would scarcely be felt, although a stack is useful to prevent the wind from baffling the current at top of the shaft. A Guibal fan 10 feet in diameter 4 feet wide would produce 15,000 cubic feet with $1\frac{1}{2}$ inches of water-gauge. The power required to move this quantity of air would be

$$\frac{15{,}000 \times 7.8}{33{,}000} = 3.54 \text{ H. P.}$$

Allowing the fan to give out 30 per cent. of its power on the air, and allowing power to increase the speed for emergencies, the fan should be driven by an engine capable of developing at least 15 horse-power.

Ques. 42.—The air course in a mine is 7×8 and 960 feet long, passing 63,480 cubic feet of air per minute; this air course is then divided into four separate splits, as follows:

First split	6×5 and	1200 feet long.
Second split	6×6 and	900 feet long.
Third split	6×4 and	840 feet long.
Fourth split	5×4 and	720 feet long.

With the same total volume, what quantity will pass in each split, and what will be the water-gauge, using Atkinson's coefficient?

Ans.—In using the formula

$$V = \sqrt{\frac{Pa}{Ks}}$$

if we denote P, the pressure, and K, the coefficient, by 1, the formula becomes

$$V = \sqrt{\frac{1 \times a}{1 \times s}},$$

then the relative velocities for each split are

$$\text{First split } \sqrt{\frac{30}{22 \times 1200}} = 0.0337,$$

$$\text{Second split } \sqrt{\frac{36}{24 \times 900}} = 0.04,$$

$$\text{Third split } \sqrt{\frac{24}{20 \times 840}} = 0.0377,$$

$$\text{Fourth split } \sqrt{\frac{20}{18 \times 720}} = 0.0392.$$

The velocities multiplied into the areas will give the relative quantities for each split:

First split $0.0337 \times 30 = 1.011$
Second split $0.04 \times 36 = 1.44$
Third split $0.0377 \times 24 = 0.9048$
Fourth split $0.0392 \times 20 = 0.784$

Total relative quantity, 4.1398

Then as the total relative quantity is to the relative quantity in each split, so is the total actual quantity to the quantity that will pass in each split.

First split 4.1398 : 1.011 :: 63,480 : 15,502.74
Second split 4.1398 : 1.44 :: 63,480 : 22,081.07
Third split 4.1398 : 0.9048 :: 63,480 : 13,874.27
Fourth split 4.1398 : 0.784 :: 63,480 : 12,021.92

Total quantity passing, 63,480

showing that the distribution is correct.

The pressure for each airway is found by

$$p = \frac{K s v^2}{a},$$

K being the coefficient for a velocity of 1000 feet per minute.

$$(1)\ p = \frac{0.0217 \times 22 \times 1200 \times 0.267}{30} = 5.08;$$

$$(2)\ p = \frac{0.0217 \times 24 \times 900 \times 0.376}{36} = 4.90;$$

$$(3)\ p = \frac{0.0217 \times 20 \times 840 \times 0.333}{24} = 5.06;$$

$$(4)\ p = \frac{0.0217 \times 18 \times 720 \times 0.362}{20} = 5.08;$$

$$\frac{5.08}{5.2} = 0.98 \text{ inch water-gauge.}$$

The slight difference in the pressure is due to the decimal fractions not being carried out far enough in extracting the roots.

Ques. 43.—Suppose a given power circulates 50,000 cubic feet per minute through an airway 6′ × 5′ and 10,000 feet long, and the ventilation is changed by dividing the airway into three different splits as follows:

First split, 6′ × 6′ and 4000 feet long.
Second split, 5′ × 6′ and 3000 feet long.
Third split, 5′ × 5′ and 4000 feet long.

What quantity will pass through each?

Ans.—These splits are subject to equal pressures and the quantities that will pass in airways of different dimensions under equal pressures, are proportional to the *square root of the result* of dividing the area by the rubbing surface, multiplied by the area, or

$$\sqrt{\frac{a}{s}} \times a;$$

therefore we will have to use this formula to distribute the total quantity among the splits according to their resistance:—

First split, $\sqrt{\frac{36}{96{,}000}} \times 36 = 0.697;$

Second split, $\sqrt{\frac{30}{66{,}000}} \times 30 = 0.640;$

Third split, $\sqrt{\frac{25}{80{,}000}} \times 25 = 0.442;$

Total relative quantity, 1.779

As the total amount of air is unknown we must assume some quantity, say 100,000 cubic feet per minute, divide it up with the splits, calculate the horse-power required to circulate that amount of air, and then correct the result to fit the horse-power which was required by the original airway.

Supposing there is 100,000 cubic feet of air in circulation, the amount in each split will be

First split = 1.779 : 100,000 :: .697 : 39,200
Second split = 1.779 : 100,000 :: .540 : 36,000
Third split = 1.779 : 100,000 :: .442 : 24,800

100,000

The velocities in the three splits will be

(1) 39,200 ÷ 36 = 1089 feet per minute.
(2) 36,000 ÷ 30 = 1200 feet per minute.
(3) 24,800 ÷ 25 = 993 feet per minute.

The pressure required by each split will be

$$(1)\ \frac{K \times 96{,}000 \times 1089^2}{36} = 68.5 \text{ lbs. per sq. foot.}$$

$$(2)\ \frac{K \times 66{,}000 \times 1200^2}{30} = 68.5 \text{ lbs. per sq. foot.}$$

$$(3)\ \frac{K \times 80{,}000 \times 993^2}{25} = 68.5 \text{ lbs. per sq. foot.}$$

All being equal as they should be.

The foot pounds per minute required to circulate 100,000 cubic feet of air per minute is:

$$68.5 \times 100{,}000 = 6{,}850{,}000 \text{ ft. lbs.}$$

Now the velocity of the original air course is:

$$50{,}000 \div 30 = 1667 \text{ feet.}$$

And the pressure required to circulate 50,000 cubic feet of air is:

$$\frac{K \times 220{,}000 \times 1667^2}{30} = 442 \text{ pounds}$$

per square foot.

The amount of work required is:

$$442 \times 50{,}000 = 22{,}100{,}000$$

foot pounds per minute.

Now it takes 6,850,000 foot pounds to circulate 100,000 cubic feet of air in the split air courses; and we must find how many cubic feet will be passed by 22,100,000 foot pounds.

As the volumes will vary as the cube roots of the works the required amount will be found as follows:

$$\sqrt[3]{6{,}850{,}000} : \sqrt[3]{22{,}100{,}000} :: 100{,}000 : 148{,}000.$$

Dividing 148,000 up between the separate splits as required by the question we have:

(1) 1.779 : 148,000 = .697 : 58,000
(2) 1.779 : 148,000 = .640 : 53,300
(3) 1.779 : 148,000 = .442 : 36,700

148,000

Ques. 44.—The downcast shaft of a colliery is

10×14 and 1000 feet deep,

the upcast or air shaft being circular, 10 feet in diameter, and the same depth. The air course under ground 6×7 and 4000 yards in length, with a volume of 40,000 cubic feet per minute passing. When the airways under ground are split as follows:

1st split 8×8 and 5000 feet long,
2d split 6×6 and 6000 feet long,
3d split 5×5 and 3000 feet long,

how much air will pass in each split, the power remaining the same?

Ans.—The areas and rubbing surfaces of the shafts and splits are:

	Area, square feet.	Surface.	Sq. ft.
Downcast . . .	$10 \times 14 = 140$	$2 \times 10 + 2 \times 14 \times 1000 =$	48 000
Upcast	$10^2 \times .7854 = 78.54$	$10 \times 3.1416 \times 1000 =$	31.416
First split . . .	$8 \times 8 = 64$	$4 \times 8 \times 5000 =$	160 000
Second split . .	$6 \times 6 = 36$	$4 \times 6 \times 6000 =$	144 000
Third split . .	$5 \times 5 = 25$	$4 \times 5 \times 3000 =$	60,000

The area and perimeter of the airway as given are $6 \times 7 = 42$ square feet, and $2 \times 6 + 2 \times 7 = 26$ feet respectively.

We must first find the foot-pounds required to pass 40,000 cubic feet of air through the original airway; next find how many foot-pounds it takes to pass any assumed quantity through the shafts and splits. We can then find the true quantity passing in the splits by a proportion between the cube roots of the powers.

The original velocities are:

Downcast $40{,}000 \div 140 = 286$ ft. per minute.
Airway $40{,}000 \div 42 = 953$ ft. per minute.
Upcast $40{,}000 \div 78.54 = 510$ ft. per minute.

The pressures per square foot required by the original arrangement are:

$$\text{Downcast } \frac{.0000000217 \times 48{,}000 \times 286^2}{140} = 0.6 \text{ lbs.}$$

$$\text{Airway } \frac{.0000000217 \times 312{,}000 \times 953^2}{42} = 146.5 \text{ lbs.}$$

$$\text{Upcast} \quad \frac{.0000000217 \times 31{,}416 \times 510^2}{78.54} = 2.3 \text{ lbs.}$$

Total pressure per sq. ft. 149.4

The number of foot-pounds of work per minute required by the air originally were:

$$40{,}000 \times 149.4 = 5{,}976{,}000.$$

The splits being subject to a *common pressure* will pass quantities in proportion to

$$\sqrt{\frac{Pa}{Ks}} \times a;$$

but as P and K are equal in all the splits, they cancel, and the formula becomes

$$\sqrt{\frac{a}{s}} \times a.$$

Then the total volumes distributed among the splits, by the above formula, are as the relative quantities passing in each:

$$\text{First split} \quad \sqrt{\frac{64}{160{,}000}} \times 64 = 1.28$$

$$\text{Second split} \quad \sqrt{\frac{36}{144{,}000}} \times 36 = .57$$

$$\text{Third split} \quad \sqrt{\frac{25}{60{,}000}} \times 25 = .51$$

Total relative quantity, 2.36

Assuming that there are 100,000 cubic feet of air passing through all the splits, the amounts passing through each will be:

(1) 2.36 : 100,000 :: 1.28 : 54,300.
(2) 2.36 : 100,000 :: .57 : 24,100.
(3) 2.36 : 100,000 :: .51 : 21,600.

100,000

In finding the total pressure required by the mine as split when passing 10,000 cubic feet of air per minute; we find the pressures for the downcast, upcast, and only one of the splits; because all the splits have equal pressures.

The velocities in the split mine are:

$$\text{Downcast } \frac{100{,}000}{140} = 715 \text{ ft. per min.}$$

$$\text{First split } \frac{54{,}300}{64} = 849 \text{ ft. per min.}$$

$$\text{Upcast } \frac{100{,}000}{78.54} = 1274 \text{ ft. per min.}$$

The pressures per square foot are:

$$\text{(D) } \frac{.0000000217 \times 48{,}000 \times 715^2}{140} = 3.8.$$

$$\text{(1) } \frac{.0000000217 \times 160{,}000 \times 849^2}{64} = 39.0.$$

$$\text{(U) } \frac{.0000000217 \times 31.416 \times 1274^2}{78.54} = 14.1.$$

Total pressure per square foot 56.9 lbs.

Work required to circulate 100,000 cubic feet per minute with mine as split is:

$$56.9 \times 100{,}000 = 5{,}690{,}000 \text{ ft.-lbs. per min.}$$

Now the volume of air circulated by 5,690,000 foot pounds per minute will be to the volume circulated by 5,076,000 foot pounds as

$$\sqrt[3]{5{,}690{,}000} = 179$$

$$\text{is to } \sqrt[3]{5{,}976{,}000} = 181, \text{ or}$$

$$179 : 181 :: 100{,}000 : 101{,}000$$

gives the total quantity; and the quantity passing in each split will be:

$$\begin{aligned} &(1)\ 179 : 181 :: 54{,}300 : 54{,}900 \\ &(2) \qquad\quad :: 24{,}100 : 24{,}300 \\ &(3) \qquad\quad :: 21{,}600 : 21{,}800 \\ &\qquad\qquad\qquad\quad\ \ 101{,}000 \end{aligned}$$

cubic feet per minute.

Ques. 45.—A fan running 200 revolutions per minute is passing 20,000 cubic feet of air, with a water-gauge of one inch. What will be the volume of air produced, and what will be the

water-gauge, when the fan is running 300 revolutions per minute?

Ans.—All other things being equal, the pressure or water-gauge, produced by a fan, will vary as the square of the speed nearly, which by the terms of this proposition is as 3 is to 2.

$$\frac{3^2 \times 1}{2^2} = 2\tfrac{1}{4}.$$

Therefore the increased water-gauge would be $2\frac{1}{4}$ inches.

The volume of the air will vary as the square root of the pressure, and the pressure varies as the square of the speed. Therefore the volume varies as the speed, or

$$\frac{3 \times 20{,}000}{2} = 30{,}000$$

cubic feet as the volume due to the increased speed.

Ques. 46.—If a power of 120,000 foot-pounds is causing a volume of air to pass through an airway $7' \times 8'$, what will be the power required to cause the same volume to pass through an airway $5' \times 5'$ both being equal in length?

Ans.—The area of the widest airway is

$$7 \times 8 = 56 \text{ square feet.}$$

Its perimeter is

$$2 \times 8 + 2 \times 7 = 30 \text{ feet.}$$

The area of the smaller airway is

$$5' \times 5' = 25 \text{ square feet.}$$

Its perimeter is

$$4 \times 5 = 20 \text{ feet.}$$

Their lengths being equal, the perimeters may be taken as their relative rubbing surfaces (s).

The volume being constant, the velocity will be inversely as the area

$$\frac{56}{25} = 2.24,$$

or the velocity in the large airway being 1, the velocity in the small airway will be $2\frac{1}{4}$. The resisting surfaces are 30 and 20 or as 3 is to 2.

The power is found by $u = k\,s\,v^3$, but k, the coefficient of friction, being equal in both, it may be cancelled, and the formula becomes

$$u = s\,v^3.$$

Using relative quantities, we get,

$$3 \times 1^3 = 3$$

for the larger airway, and

$$2 \times 2.24^3 = 22.48$$

for the smaller airway, as relative powers. Then by proportion,

$$\frac{22.48 \times 120{,}000}{3} = 899{,}200$$

foot-pounds, or

$$\frac{899{,}200}{33{,}000} = 27.25$$

horse-power will be required by the small airway for an equal quantity.

Ques. 47.—While the fan is running at the same speed, will a larger quantity of air pass through an airway $8' \times 10'$, than through one $6' \times 8'$?

Ans.—Yes. Because the large airway has less resisting surface in proportion to its area.

If the question means that the fan is acting on both airways at the same time, then they are splits of the same total volume, and are subjected to equal pressure, and will pass quantities according to the formula

$$q = \sqrt{\frac{p\,a}{k\,s}} \times a.$$

But in this case the pressure p and the coefficient k are equal for both, therefore the formula becomes

$$q = \sqrt{\frac{a}{s}} \times a.$$

No length being given, the perimeters will have to be taken as the relative surfaces, s.

The larger airway is $8 \times 10 = 80$ square feet area, and has $2 \times 8 + 2 \times 10 = 36$ feet of perimeter.

The smaller airway is $6 \times 8 = 48$ square feet area, and has $2 \times 6 + 2 \times 8 = 28$ feet of perimeter. Therefore,

$$\sqrt{\frac{80}{36}} \times 80 = 119.2$$

is the relative quantity passing in the large airway, and

$$\sqrt{\frac{48}{28}} \times 48 = 62.4$$

the relative quantity passing in the small airway.

This shows that the quantity for the large airway is to the quantity for the small one, as 1.91 is to 1, or practically nearly double, with the fan at any speed. If the question means, as we would incline to interpret it, that the fan at a constant speed acts on each airway separately for comparison, then the resulting difference is not so great. Because the pressure per foot, at the fan, as indicated by the water-gauge, is a function of the speed, it is constant with a constant speed, less what is lost by increase of friction, as larger volumes pass through the fan.

When an increased volume passes, with the fan at a constant speed, the water-gauge is reduced by a quantity dependent on the construction or "characteristic curve of the ventilator," and can only be determined from known quantities. Approximately a constant speed produces a constant pressure, and because the pressure at the point near, or at the fan, multiplied by the quantity passing is the amount of *power*. Therefore the steam pressure must be increased in proportion to the quantity of air passing in order to maintain the fan at a constant speed, which, in this case, is an increase of power from 1 to 1.91.

Ques. 48.—With a fan at constant speed, on airways of equal length, which would pass the largest quantity of air; four airways $4' \times 5'$, or one airway $8' \times 10'$?

Ans.—The largest quantity would pass in the one airway $8' \times 10'$; because the area of the four is equal to the area of the one, and their resisting surfaces are much greater. The areas being equal makes the pressure equal, and the quantity is inversely as the square root of the surface. The sum of the areas of the four small airways is

$$4 \times 4 \times 5 = 80 \text{ square feet,}$$

and the sum of their perimeters is

$$(2 \times 4 + 2 \times 5) \times 4 = 72 \text{ feet.}$$

The area of the large airway is

$$8 \times 10 = 80 \text{ square feet,}$$

and its perimeter is

$$2 \times 8 + 2 \times 10 = 36 \text{ feet.}$$

The areas are equal, but the perimeters are as 2 is to 1.

The quantity in both cases is according to

$$q = \sqrt{\frac{p\ a}{k\ s}} \times a,$$

but the total pressure (Pa), the coefficient (K), and the area (a) are equal in both cases.

Therefore they cancel, and the formula becomes

$$q = \sqrt{\frac{1}{s}}$$

Substituting the known relative quantities, we obtain

$$\sqrt{\frac{1}{2}} = 0.707$$

as the relative volume which would pass in the four small airways, and

$$\sqrt{\frac{1}{1}} = 1$$

would be the relative quantity that would pass in the one large airway,

$$\frac{1}{0.707} = 1.41$$

Therefore the volume passing in the large airway would be to the volume passing in the four small airways as 1.41 is to 1, but the fan would require to give out $\frac{1}{2}$ more power to maintain the constant speed, which would be $\frac{1}{2}$ more pressure of steam, the piston speed being constant.

Ques. 49.—If 10,000 cubic feet of air per minute pass through an airway $7' \times 8'$, what quantity will pass, with the same power, through one $2' \times 5'$?

Ans.—Airways, under equal power, will pass quantities di-

rectly as their areas and inversely as the cube root of their surfaces, or as the cube root of the reciprocal of their surfaces into their areas, or as

$$\sqrt[3]{\frac{1}{s}} \times a.$$

In this problem no length is given, therefore the perimeter may be taken as the surface (s).

The perimeter and area of the large airway are

$2 \times 7 + 2 \times 8 = 30$ feet, and $7 \times 8 = 56$ square feet, respectively, and for the small airway they are

$4 \times 5 = 20$ feet, and $5 \times 5 = 25$ square feet, respectively.

$$\sqrt[3]{\frac{1}{30}} \times 56 : \sqrt[3]{\frac{1}{20}} \times 25 :: 10{,}000 : 5153+$$

cubic feet per minute as the quantity that would pass in the small airway under the given conditions.

Ques. 50.—If it requires 8 horse-power to pass 20,000 cubic feet of air per minute through a mine, the pressure being 5 pounds per square foot, how many horse-power will be required to pass 25,000 cubic feet per minute with the same pressure?

Ans.—To pass 25 thousand cubic feet will require much more power than to pass 90 thousand, but it will also require more pressure, except some change is made in the mine. Either the surface (s) must be reduced by a shorter circuit for the current, or a lower velocity (v) by a larger area (a), so that a larger volume and greater power can be applied without change of pressure; because the pressure into the quantity is the power (u), and the pressure is constant, therefore to obtain more quantity and apply more power some other factor must be reduced. If a door is opened in a mine, the pressure at the fan is reduced, but more air comes to the fan and the fan speed is decreased, and if more steam is turned on, and the fan speed is increased until the pressure is the same as before, then there is more air, and more power with the same pressure, the opening of the door having reduced the extent of the rubbing surface s. The same result would follow by widening the return airway. This question was asked by a board of examiners, and is not clearly stated, or is intended as a catch. Assuming some change in the

mine so as to allow of equal pressure, we get the following solution:—

Because (p) the pressure into (q) the quantity is equal to (u) the power, and because (p) is made equal in both cases

$$u = pq \text{ becomes } u = q,$$

or the quantity is according to the power, which by proportion will be

$$20 : 25 :: 8 : 10$$

horse-power as the power for 25,000 cubic feet.

Ques. 51.—Suppose a mine has two openings, one 60 feet higher surface elevation than the other, with the temperature outside of 85° and inside of the mine 50°, would a current be produced, and, if so, in what direction?

Ans.—The difference of height and temperature is so limited that their combined effect would not produce motion amounting to a *current*. Under a barometric pressure of 30 inches, air at 50° weighs 0.0781113, and at 85° 0.0730858 of a pound per cubic foot, and 0.0781113—0.0730858 = 0.0050255 of a pound as their difference per cubic foot; the vertical difference in height is 60 feet, therefore the pressure per foot is 0.0050255 × 60 = 0.30153 of a pound, equal to a water-gauge of about

$$\frac{0.30153}{5.2} = 0.0579 \text{ of an inch,}$$

a pressure too light to be *practically* felt in the average mine.

Still, because the outside temperature is the higher, the tendency to motion would be into the mine by the higher and out by the lower opening.

The weight per foot of air is found by

$$W = \frac{1.3253 \times B}{459 + T}$$

where W is the weight per cubic foot in fraction of a pound, B the height of the barometer in inches, and T the temperature of the air (Fahr.).

The above problems have by no means exhausted the subject of ventilation, but we think that their solution by the student will make him sufficiently acquainted with the principles involved to qualify him for all practical purposes connected with the subject.

CHAPTER V.

VENTILATION, CONTINUED.

Ques. 1.—Explain the principle of the ventilating fan.

Ans.—The main principle involved in the ventilating fan is "centrifugal force." When a boy slings a stone he whirls it around until it attains a high velocity; the tendency to fly outwards from the centre of rotation is measured by the tension of the string. When the stone is released this tension or force is converted into velocity, throwing the stone to a great distance. The work thus stored up in the stone is the "centrifugal force."

When a fan revolves at a considerable speed, the air carried round between the vanes, having weight, is thrown out from the tips of the vanes by the same force which throws the stone from the sling. This displacement of the air from the centre produces a partial vacuum, and the difference of pressure causes air to rush in at the centre to replace what is thrown out at the periphery.

For the purpose of illustrating its application as a ventilator we will suppose a mine connected with the surface by two shafts. On the assumption that the temperature throughout the mine and shafts is nearly equal, the one shaft will evidently balance the other, and there will be no motion. If a fan is placed near and its centre connected to one of the shafts, as the fan revolves the air thrown out at the periphery will be supplied from the mine. If the fan draws 100,000 cubic feet per minute from the shaft, the same quantity will flow into the other shaft to maintain the level, like water or any other fluid.

Let us now close the other shaft, so that no air can get down. If the fan now runs until half a million cubic feet have been drawn from the mine, it is evident that it has lost just that much weight, and is that much lighter, bulk for bulk; but the mine is still full. Why? Because the air has expanded. When the

fan has exhausted air until the difference of the pressure, or rather weight, of the air inside and outside is equal to the centrifugal force of the fan, then the current from the mine will cease and the fan merely whirl around without any apparent action. Under these conditions, although the fan is not drawing air, it is spending force in keeping the outside air from backing through the fan into the shaft. The fan is now maintaining pressure, but doing no work. It is like a steam-engine held by the break, with full steam on—the pressure is there, but no power is being developed; there is the force, but time and space are wanting to constitute power. If the cover is now suddenly removed from the other shaft, the effect is to check the speed of the fan, requiring more steam on the engine to maintain the same number of revolutions per minute, because the air in passing through the fan creates resistance, which increases as the fan is traversed by larger volumes.

Where the air has to travel long distances at high velocities the frictional resistance in the mine becomes very great and prevents the air from flowing to the fan as fast as the fan would throw it out; it therefore expands to make up the difference, and this difference is the pressure overcoming the frictional resistance and creating the velocity. When the fan is applied as a "blower," the periphery is enclosed and connected to the mine and the centre open to the atmosphere, and thus reverses the action by blowing into instead of exhausting out of the shaft, making the difference of pressure above instead of below the atmosphere. The difference between ventilating with a fan and a furnace is in the mode of disturbing the equilibrium; the furnace causes it by expansion from heat, while the fan causes it by expansion from displacement. When the fan is placed inside of the mine, as at some collieries in the anthracite region, the principle is the same; the fan displacement in the one direction causes the air to flow in from the other to maintain the balance.

This is only the effect the fan produces; but to follow the air as it traverses the fan and make an analysis of its behavior while passing through is a very different problem, indeed. Nearly every gaseous mine is ventilated by a fan, but it is ques-

tionable if every mine has a manager who understands the simple-looking machine which supplies it with air.

The "Transactions" of mining institutes and kindred societies contain many valuable papers on this very interesting subject; but those that I have seen are demonstrated by mathematical formulæ wholly unintelligible to the average miner. There are different types of fans in use, and every inventor claims superiority for his own. The results given by experimenters with different fans are so contradictory as to be very unsatisfactory. The difference of construction shows a difference of opinion as to the principles involved; and when we consider that the inventors of the different fans are men acquainted with "mechanics," we naturally infer that there is something about centrifugal ventilators not easily understood.

The difficulty in making fan experiments is that no two are working under exactly the same conditions. At one mine a Guibal fan 18 feet diameter, running 70 revolutions, may be passing 60,000 cubic feet per minute; while at another mine a Schiele fan only 12 feet diameter, running 90 revolutions, may be passing 100,000 cubic feet per minute. The Guibal may be raising a water-gauge of $1\frac{1}{3}$ inches, while the Schiele has only 0.7 of an inch; and if the fans were exchanged, both might give worse results.

The Guibal suits best for small openings, long airways, and heavy "drag;" while the Schiele is best adapted to large airways, split currents, and moderate "drag."

M. Murque, a French engineer, has investigated the theory of fans by a method which appears to surmount some of the difficulties encountered by other investigators, and overcomes this difficulty by what he terms the "equivalent orifice," which can perhaps be best explained by working an example.

Suppose two fans, placed in adjacent mines, one passing 100,000 cubic feet per minute with 3 inches of water-gauge, the other passing 75,000 feet with a water-gauge of 4 inches. The power expended on the current for the first fan is

$$3 \times 5.2 \times 100{,}000 = 1{,}560{,}000; \frac{1{,}560{,}000}{33{,}000} = 47 \text{ H. P.};$$

and for the second fan it is

$$4 \times 5.2 \times 75{,}000 = 1{,}560{,}000; \quad \frac{1{,}560{,}000}{33{,}000} = 47 \text{ H. P.}$$

The power expended being equal in both cases, if it is found that the engine of the first fan is developing 10 or 20 horse-power more than the engine of the second, it will be difficult to compare them for the purpose of determining what caused this absorption of the extra power.

If we were now to obstruct the first mine until the volume was reduced to 75,000 feet and the water-gauge increased to 4 inches, the two fans would be working under similar conditions; but the difficulty would be to get both to the exact point for comparison.

Assuming 1.2 and 1000 to be the relative densities of air and water, or that water is 833 times heavier than air under ordinary conditions, the first fan's water-gauge of 3 inches would be equal to a column of air $3 \times 833 = 2499$;

$$\frac{2499}{12} = 208 \text{ feet high.}$$

This column falling freely through its own height would attain a velocity of

$\sqrt{208} = 14.42$ and $14.42 \times 8 \times 60 = 6921.6$ feet per minute. This is the velocity the first fan pressure would generate without resistance.

The second fan's water-gauge of 4 inches would be equal to a column of air $4 \times 833 = 3332$;

$$\frac{3332}{12} = 277.65 \text{ feet high,}$$

and the velocity due to this height is $\sqrt{277.65} = 16.6$ and $16.6 \times 8 \times 60 = 7968$ feet per minute, as the theoretical velocity.

If the volumes passed by each fan are now divided by the theoretical velocity due to their pressures, it will show the area of an orifice having no resistance through which the given pressure would force the given volume in each case.

Experiment proves that when air is forced through an orifice in a thin sheet, having practically no sides to cause friction, that only about $\frac{2}{3}$, or 65 per cent. of the quantity due to the

area and velocity pass; therefore if we add 54 per cent. to the volume, or take 65 per cent. of the velocity, it will give us the area of the hole or orifice in a thin board through which the pressure would force the quantity. For the first fan this would be

$$6921.6 \text{ feet velocity} \times 0.65 = 4499.04.$$

and

$$\frac{100{,}000}{4499.04} = 22.22 \text{ square feet,}$$

and for the second fan it would be

$$7968 \text{ feet velocity} \times 0.65 = 5179.2$$

and

$$\frac{75{,}000}{5179.2} = 14.44 \text{ square feet.}$$

These are what are called the "Equivalent Orifices" of each mine respectively, which may be found by the following formula:

$$a = \frac{0.000385 \times q}{\sqrt{w}}$$

Where a = Equivalent Orifice in square feet,
Where q = Quantity of air in cubic feet per minute.
Where w = Height of water-gauge in inches.

If the mine is now closed off from the fan by a stopping, and a hole cut in the side of the fan drift equal to the Equivalent Orifice of the other mine, then we could test what the one fan could do in the other's place, and make comparison. Orifices can thus be calculated for any kind of mine when the pressure and quantity of air passing are known.

M. Murque's second proposition is that every fan while at work offers resistance to the air passing through it, which resistance increases as larger volumes traverse the fan and absorbs a corresponding fraction of the initial pressure due to its peripheral velocity. M. Murque also represents this resistance by another orifice, which he calls the "Orifice of Passage." His third and perhaps most important proposition is that: In a thoretically perfect fan the depression due to the peripheral velocity is equal to a column twice the height necessary to generate such velocity in a falling body, or

$$H_0 = \frac{v^2}{g}.$$

This is on the assumption that the fan is closed off from the mine, and that there is neither shock, leak, nor loss of any kind.

This "Ideal" fan cannot be realized in practice, because of imperfections, and is only assumed for the purpose of demonstrating the theory. The application of this formula can, perhaps, be best illustrated by working an example.

Let us suppose our first fan, that passed 100,000 cubic feet, with three inches of water-gauge, to be a Guibal, 20 feet diameter, making 100 revolutions per minute; then, according to M. Murque, the theoretical depression in air column would be

$$\frac{20 \times 3.1416 \times 100}{60} = 104.72 \text{ feet per second}$$

as the peripheral velocity. To generate such velocity requires a height of

$$\frac{104.77^2}{64.4} = 170.283 \text{ feet.}$$

This doubled would be an air column of $170.283 \times 2 = 340.566$ feet as the height theoretically due to the fan at that speed. This column in falling through its own length would attain a velocity of

$$\sqrt{340.566} \times 8 \times 60 = 8856 \text{ feet per minute.}$$

Taking 65 per cent. of this velocity and dividing the volume of air by it as before, will give a theoretical Equivalent Orifice equal to $8856 \times 0.65 = 5756.4$ and

$$\frac{100{,}000}{5756.4} = 17.37 \text{ square feet.}$$

This is the size of orifice through which the theoretical pressure would force the quantity. But from imperfections it can only force it through an orifice of 22.22 feet, as before shown;

$$\frac{22.22}{17.37} = 1.27.$$

Therefore the actual result is to the theoretical result as 1 is to 1.27. This shows that the fan theoretically should force the air through an orifice 17 feet, but practically it requires an orifice 22 feet.

M. Murque starts out to investigate his "Ideal" fan on the

assumption that it is theoretically perfect, and that the air which passes through is incompressible within the limits of practice. When we consider that the highest pressures produced by fans, either positive or negative, will not exceed 3 or 4 inches of water-gauge, we may assume 2 inches as the average. The total pressure of the atmosphere is about 32 feet. 32 feet $\times$ 12 = 384 inches, and

$$\frac{384}{2} = 192.$$

Therefore the average water-gauge will not change the bulk of a given weight of air more than $\frac{1}{192}$, a difference which may be ignored in order to avoid complications in the investigation.

For the purpose of investigating the theoretical efficiency of the fan in regard to the depression it will produce, we will suppose the fan revolving at a constant speed, and closed off from the mine by a stopping. The fan, though producing no current, will maintain a certain water-gauge, which may be termed the *initial depression*, or head of air column, H. If an orifice is now made in the stopping, the effect will be to lower the depression, hence we reason *that the depression will be reduced to zero when the Equivalent Orifice is increased to infinity.* Supposing the air discharged from the fan had to pass through another orifice which was equivalent to the resistance encountered by the air in passing through the fan, then the resistance of the mine and the fan would be represented by the two orifices respectively. In these investigations M. Murque uses the following notation:

H = The initial depression in feet of air column.
h_0 = The loss of depression from resistance in the fan.
h = The effective depression in feet of air column.
a = The Equivalent Orifice of the mine.
o = The Equivalent Orifice of the fan.
V = The volume of air passing in cubic feet per second.

The velocity due to the height of column, and the column generative of the velocity, is represented by the usual formula for gravity—

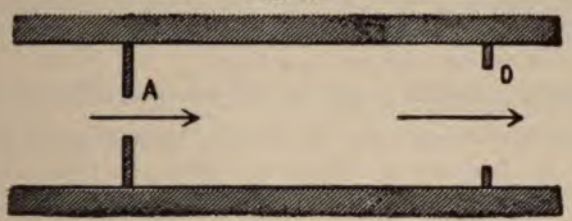
$$h = \frac{v^2}{64.4}, \text{ or } v = 8.02\sqrt{h} = \sqrt{64.4\,h} = \sqrt{2\,g\,h}.$$

The effective depression h will therefore be

$$h = 'H - h_0, \qquad (1)$$

h_0 being the depression absorbed by loss in the fan, which may increase from zero up until it absorbs H, when the orifice is in-

FIG. 31.



creased to infinity. In Fig. 31 the mine orifice A and the fan orifice O being traversed by the same volume, their corresponding depressions are in inverse proportion to their squares, or

$$\frac{h_0}{h} = \frac{a^2}{o^2}. \qquad (2)$$

By substituting for h_0 in the first equation the value resulting from the above proportion, the next equation obtained is

$$h = H - h \times \frac{a^2}{o^2}, \qquad (3)$$

which reduced in the proportion to h gives the first formula required, which becomes

$$h = \frac{H}{1 + \frac{a^2}{o^2}}. \qquad (4)$$

The given volume V, by the formula for air flowing through an orifice having practically no sides to cause friction, is, as before stated, equal to

$$V = 0.65\ o\ \sqrt{2\ g\ h_0}.$$
$$V = 0.65\ a\ \sqrt{2\ g\ h}. \qquad (5)$$

Replacing h by the preceding value or its equivalent in equation 4, it becomes

$$V = \frac{0.65\ a\ \sqrt{2\ g\ H}}{\sqrt{1 + \frac{a^2}{o^2}}}. \qquad (6)$$

The Equivalent Orifice of the mine a is supposed to be known. The orifice of passage depends on the dimensions and arrangement of the fan. The depression and volume of air produced by the fan are therefore expressed by equations 4 and 6.

For the purpose of illustrating these equations by known quantities, let us take the fan used in our former example.

20 = Diameter in feet.
100 = Revolutions per minute.
340.566 = Height in column due to speed of fan = H_0.

The work performed by the fan was 100,000 cubic feet per minute with 3 inches of water-gauge, giving an Equivalent Orifice of 22.22 square feet for the mine, the 3 inches of water column being equal to a column of air.

$$\frac{3 \times 833}{12} = 208 = h = \text{the effective depression.}$$

The initial depression is from $\frac{1}{2}$ to $\frac{3}{4}$ of the theoretical depression according to the make of the fan. The initial head in our example is say $\frac{3}{4}$ of 340.566 = 255 feet = H.

If we take from the initial depression H the effective depression h, it will leave the amount of depression h_0 lost in the fan.

$$H = 255$$
$$h = 208$$
$$h_0 = 255 - 208 = 47 = h_0,$$

as the height of depression lost in the fan. The velocity due to such height is

$$\sqrt{47} \times 8 \times 60 = 3290 \text{ feet per minute.}$$

The Equivalent Orifice for such velocity with 100,000 cubic feet per minute is equal to $3290 \times 0.65 = 2140$, and

$$\frac{100{,}000}{2140} = 46.8 \text{ square feet,}$$

as the orifice of passage through the fan = o.

Let us now find the value of the orifice of passage by the proportion given in equation (2).

$$\frac{h_0}{h} = \frac{a^2}{o^2} = h_0 : h :: a^2 : o^2.$$

By substituting the known quantities in the above proportion we will find the value of the orifice of passage o.

$$47 : 208 :: 22.22^2 : 46.8^2,$$

which is the same value as found by taking the velocity and from it calculating the orifice of passage, which was found to be 46.8 square feet. This proves equation 5 to be correct.

Let us now try to prove equation 6 by known quantities.

$$V = \frac{0.65\, a \sqrt{2\, g\, H}}{\sqrt{1 + \frac{a^2}{o^2}}} = \text{volume per second.}$$

$$V = \frac{0.65 \times 22.22 \times \sqrt{64.4 \times 255}}{\sqrt{1 + \frac{22.22^2}{46.8^2}}} =$$

$$\frac{1845}{1.109} = 1666 \text{ cubic feet per second,}$$

and

$$1666 \times 60 = 100{,}000 \text{ cubic feet per minute}$$

as the effective volume V. The actual volume given by the fan was 100,000 feet per minute. The above result proves equation 6 to be correct.

Having laid down his *fiction* of the Equivalent Orifice and established the preceding formulæ, M. Murque proceeded to determine the Initial Depression produced by his ideal fan, first as an open running fan, and second with the application of a spiral casing, terminating in an expanding chimney, as now generally used in the anthracite coal region.

For the purpose of rendering M. Murque's demonstration more simple we will take the fan used in our former example, and by taking the known quantities from the fan, attempt to illustrate M. Murque's formula as we proceed.

His proposition that the theoretical depression due to the peripheral velocity is equal to twice the height generative of such velocity, is demonstrated by using the following notation:

r = radius of inlet in feet = 5 feet.

R = radius of fan in feet = 10 feet.

w = angular speed of rotation = 10.472.

u = peripheral velocity in feet per second = 104.72 feet.
g = 32.2 the force of gravity.
H_0 = theoretical depression in air column = 340.566 feet.

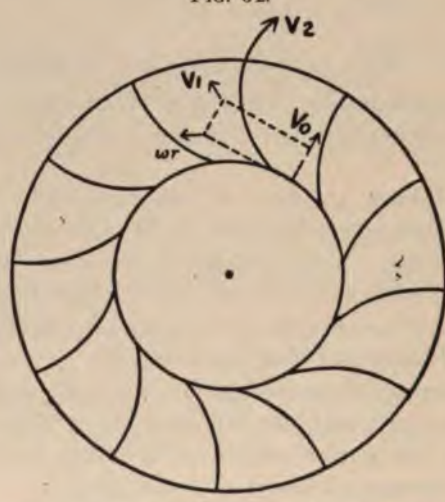
V_0, V_1, V_2 represent different velocities in the fan, and their values will have to be found as we proceed with the analysis.

The fan is 20 feet in diameter, making 100 revolutions, passing 100,000 cubic feet per minute with 3 inches water-gauge = 208 feet of air column.

We will assume the fan without cover or chimney to be represented by Fig. 32.

The vanes are shown as starting nearly tangential from the circumference of the inlet. This is to allow the air to enter without shock, and they are sufficient in number and close enough to guide the current steadily and without re-entries or eddies to cause disturbance in the interior of the fan.

FIG. 32.



The depression produced by such a fan is the result of different actions, some additive and some subtractive. These actions must be determined one after another, and placed + or — according as they may be positive or negative.

In Fig. 32 the dotted lines are intended to show the direction of the forces.

First. When the air enters it is brought to the speed V_0, with which it enters in between the vanes. The height necessary to generate this speed has to be taken from the initial depression. It is, therefore, a negative quantity, and its value is

$$-\frac{V_0^2}{2g},$$

and the known quantity for the fan with opening on one side 10 feet diameter is

$$10^2 \times 0.7854 = 78.54 \text{ square feet area and}$$

$$\frac{100{,}000}{78.54} = 1273.23,$$

$$\frac{1273.23}{60} = 21.22 \text{ feet per second} = V_0;$$

the height or depression generative of this speed is

$$\frac{21.22^2}{64.4} = 6.9919 \text{ feet.}$$

Therefore

$$\frac{V_0^2}{2g} = 6.9919 \text{ feet of depression.}$$

Second. When the air is drawn in between the vanes it has two separate motions, one movement relative to the dragging along the surface of the vanes, and another movement produced by the rotation of the fan. Now, theoretically, the total depression is the sum of the depressions due to both of the motions.

In examining the first of these two movements we find the space between every two vanes forms an expanding passage, which the air enters with a certain velocity V_1, and leaves with a reduced velocity V_2; therefore, according to Bernouilli's theory, this slowing action produces an increase of pressure in proportion to the difference between the speeds, which is expressed by the difference of the heights of column generative of the respective velocities, or

$$\frac{V_1^2}{2g} - \frac{V_2^2}{2g}.$$

Now because the radius of the inlet is 5 and that of the fan 10 feet, therefore the area of each space between the vanes at the point of discharge is double that at where it enters. Double the

be equal to a height of 170.283 feet, and the negative quantity

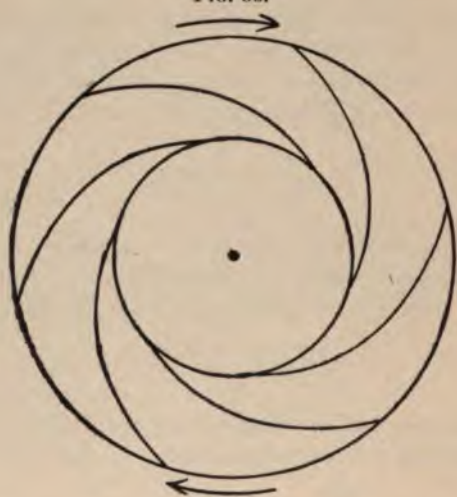
$$-\frac{V_2{}^2}{2g},$$

representing the speed of expansion due to the volume passing through between the vanes was found to be 12.4. If we now take this value from the depression due to the peripheral velocity, the remainder should be the actual depression produced

$$170.383 - 12.4 = 157.883 \text{ feet},$$

which is the same depression as found by our analysis nearly.

FIG. 33.



This result shows that the theoretical depression produced by an open fan is equal to the height generative of the peripheral velocity less the height generative of the speed with which the air leaves the periphery, due to the area of the space between the tops of the vanes, which speed must not be confounded with the tangential speeds of the vanes (u).

The depression produced by an open fan must therefore be always much less than the height generative of the speed of the periphery. The open fan complies with the first condition to produce good effect, as the air enters without shock; but it fails

to comply with the second, to leave without speed. The air from the fan is thrown into the atmosphere at a high speed, resulting from the peripheral velocity (u) and the speed of outlet V_2 due to the volume and area of passage through the spaces between the vanes at the periphery.

Constructors of fans have tried to remedy this defect by bending back the tips of the vanes, as shown in Fig. 33, so as to be nearly tangential to the periphery. This, of course, reduced the speed of outlet, but the result has not been satisfactory. It opposes the relative speed of escape to the speed of the tips of the vanes, and weakens the absolute speed of expulsion by opposing the speed V_2 to the speed u. Now before u can be effectively neutralized, V_2 must be greatly increased. But to increase V_2 increases the negative term

$$-\frac{V_2^2}{2g}$$

and decreases the head generated by the fan If the speed is increased, the passive resistance of the whole machine is increased, and what may be gained in the one way is lost in the other.

Let us now envelop the fan in a spiral case or cover terminating in an expanding chimney, such as is used in the anthracite region, and shown in Fig. 34.

The air, instead of being thrown out against the back pressure of the atmosphere all around the surface of the periphery, is discharged through the narrow passage AB, which is continued into the expanding chimney W.

The speed of the air after passing AB is slowed in proportion as the area of the chimney is increased, and the air is expelled into the atmosphere at a velocity low enough to be neglected. This reduction of speed, according to Bernoulli's theory, produces an increase of pressure to be added to the depression produced by the fan.

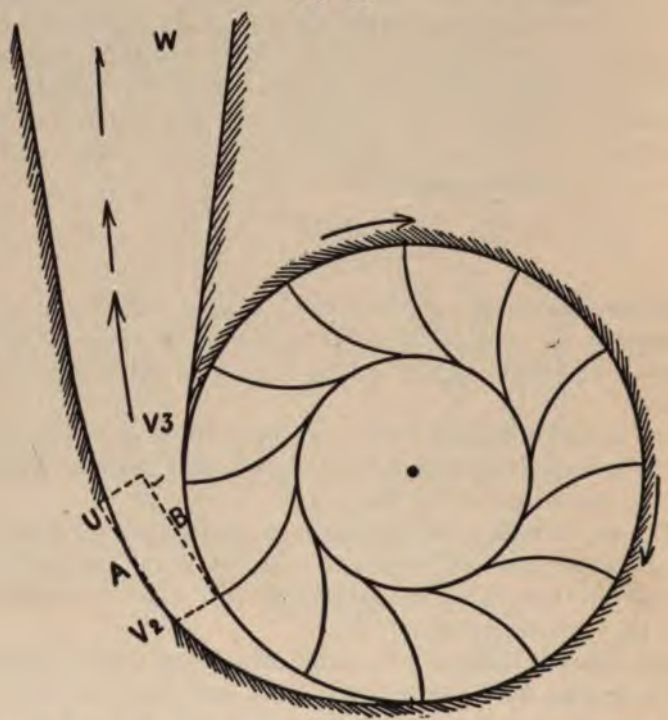
In order to illustrate this principle, let Fig. 35 represent an expanding chimney or airway, with a current of air passing in at A and escaping into the atmosphere at B. The area where the air enters is $5 \times 5 = 25$ square feet, and where it escapes

the area is increased to $10 \times 10 = 100$. Suppose the volume to be 10,000 cubic feet per minute, then the velocity of entry at *A* will be

$$10{,}000 \div 25 = 400 \text{ feet per minute,}$$

and where it escapes the velocity is reduced to $10{,}000 \div 100 =$ 100 feet per minute. Now it is evident that when the air enters

FIG 34.



at *A* there is force enough stored up in it to produce a velocity of 400 lineal feet per minute, but where it leaves there is only enough to move it at 100 feet. There has been a force expended somewhere able to generate a velocity of 300 feet. Except what is absorbed in friction, it has not gone into the sides of the box, therefore it must have been converted into pressure at the point of escape into the atmosphere, and the pressure gained is equal

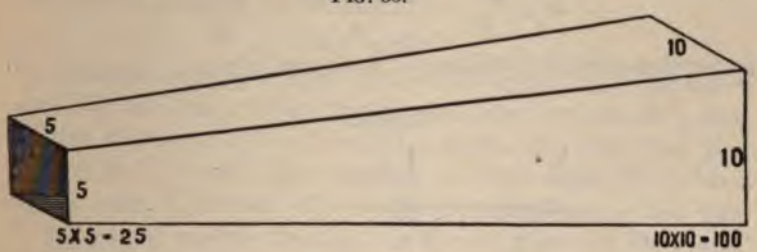
to the speed lost, less what is spent on friction. This is the theory which led to the application of the expanding chimney to fans.

If V_3 is the speed at the bottom of the expanding chimney (Fig. 34), and W the speed at the top, then, according to theory, the pressure recovered at the point of escape will be equal to the height generative of their difference of velocity, or

$$+ \frac{V_3^2}{2g} - \frac{W^2}{2g}.$$

The velocity V_3 is that with which the air leaves the tips of the vanes. It is, therefore, the resultant of the peripheral velocity u and the speed of outlet through fan V_2, and the

FIG. 35.



parallelogram of these two speeds gives the known relation between them

$$V_3^2 = u^2 + V_2^2 - 2uV_2 \text{ cosine } a,$$

where a is the angle at which the vanes strike the periphery.

Now if V_3^2 is the resultant of u and V_2, we may substitute them for its value in the preceding expression, and it will become

$$+ \frac{u_2}{2g} + \frac{V_2^2}{2g} - \frac{2\,u\,V_2 \cos.\, a}{2g} - \frac{W^2}{2g}.$$

But the fan without the cover has already produced a depression of

$$+ \frac{u^2}{2g} - \frac{V_2^2}{2g}.$$

If we add these values together, their sum will be the different

gains and losses, and by cancelling the equal plus the minus quantities the remainder will be the net result, as follows:

$$+\frac{u^2}{2g}-\frac{V_2{}^2}{2g}+\frac{u^2}{2g}+\frac{V_2{}^2}{2g}-\frac{2\,u\,V_2\cos.\,a}{2g}-\frac{W^2}{2g}=$$

$$+\frac{u^2}{2g}+\frac{u^2}{2g}-\frac{u\,V_2\cos.\,a}{g}-\frac{W^2}{2g}=$$

$$+\frac{2u^2}{2g}-\frac{u\,V_2\cos.\,a}{g}-\frac{W^2}{2g}=$$

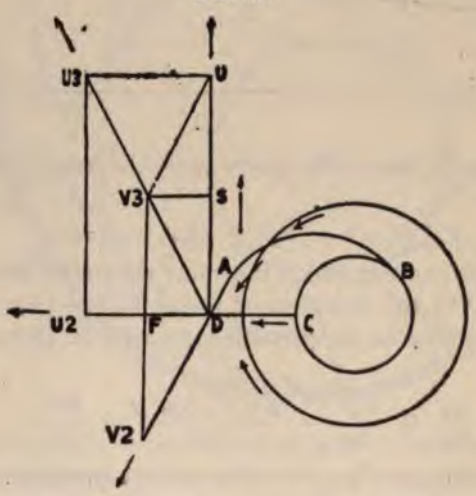
$$+\frac{u^2}{g}-\frac{u\,V_2\cos.\,a}{g}-\frac{W^2}{2g}.$$

Leaving as a final depression H produced by the fan, the expression

$$H=\frac{u^2}{g}-\frac{u\,V_2\cos.\,a}{g}-\frac{W^2}{2g}$$

as developed by the addition of the spiral cover.

FIG. 36.



In order to illustrate how bending back the vanes to an angle a with the periphery affects the diagonal of the parallelogram of the two speeds uV_2, let BA (Fig. 36) represent the vane bent back, striking the periphery at an angle $A = a$. If the lines DU

and DV_2 represent in magnitude and direction the velocities u and V_2, respectively, then the diagonal DV_3 of the parallelogram DV_2V_3U will be the resultant of the two forces or velocities.

Again, let CD be another vane which is radial, striking the periphery at an angle of 90°. If DU_2 and DU represent in magnitude and direction the velocities V_2 and u, respectively, then the diagonal DU_3 of the parallelogram DU_2U_3U will be the resultant. The difference in value of the diagonals shows the effect of bending back the vanes.

Geometrical demonstration proves that the square of DV_3 is less than the square of DU_3, by twice the rectangle under DU and DU_2, multiplied by the cosine of the angle A, which is the negative quantity represented by

$$-2\ uV_2 \text{ cos. a.}$$

Let the angle A be 30°, then because the angle DV_2F is on the same side of a line crossing two parallel lines it is also 30°, and the line FD is $\frac{1}{2}$ of the line $D\ U_2$; again, assuming that the angle $U\ D\ U_3$ is 30°, the triangles on both sides of $D\ F$ are *similar*, and V_2F and FV_2 are equal, and because the line $V_2\ V_3$ passes square through the middle point of DU_2 and meets the diagonal at V_2 it bisects it, and the parallelogram DFV_3S is only $\frac{1}{4}$ of the parallelogram DU_2U_3U.

Substituting known quantities for these values, let

$$DU = 10.4,$$

$$DV_2 \text{ or } DU_2 = 6;$$

then the value of the diagonal DU_3 is

$$10.4^2 + 6^2 = 144.76,$$

$$\sqrt{144.76} = 12.03;$$

and the diagonal DV_3 being only $\frac{1}{2}$ of DU_3 is

$$\frac{12.03}{2} = 6.01;\ 6.01^2 = 36.12.$$

The difference of these two squares is the negative quantity

$$144.76 - 36.12 = 108.64 = 2\ uV_2 \text{ cos. a.}$$

Substituting the known quantities for the above expression, we have

$-2uV_2$ cosine a$=108.64 = -2\times10.4\times6\times0.86602 = 108.08$, which proves the proposition to be correct within the limits of fractional error.

The same result can be demonstrated geometrically for any angle A.

When the vane is radial, or 90°, the resultant is the full value, and the negative quantity,

$$-2\ uV_2 \text{ cos. a},$$

disappears, as shown by M. Murque, because there is no cosine V_2F to draw down the parallelogram and reduce the diagonal. The great difference in effect caused by the cosine is due to the fact that the quantities represented are not the diagonals, but their squares, which are the equivalent of the sum of the squares of the respective velocities.

Cancelling the negative quantity leaves for the theoretical quantity the value

$$H = \frac{u^2}{g} - \frac{W^2}{2g}.$$

If we now imagine the chimney prolonged until the current, as it meets the outside atmosphere, has its velocity reduced to zero, then the value of W will be zero, and there only remains the simple quantity representing the theoretical depression produced by the Ideal fan, equal to

$$H_0 = \frac{u^2}{g},$$

which means that the depression produced by a theoretically perfect fan is equal to twice the height necessary to generate the peripheral velocity.

Such is the method by which M. Murque demonstrates his theories of the equivalent orifice and the initial depression.

The result of this investigation would lead us to the following conclusions:—

First. That the open fan first introduced at our mines, if theoretically correct, would produce a depression equal to the height generative of the speed of the tips of the vanes less the height generative of the speed of expulsion due to the volume and area of passage through the fan between the outer ends of the vanes.

Second. That the application of the cover and expanding chimney, by reducing the velocity and converting part of it back into pressure, increases the effect. The power is first expended to produce the velocity; the velocity produces the partial vacuum or exhaustive power. Then, in order to overcome the back-pressure of the atmosphere, the air is discharged through the narrow passage at the bottom of the chimney, on nearly the same principle that an injector forces water into a boiler, and by being slowed down nearly to zero in the chimney the velocity is changed to pressure, to prevent the outside atmosphere from backing into the fan.

FIG. 37.

Third. That the vanes should be nearly tangential where the air enters, so as to prevent shock, and curve round to the direction of the radius, so that the angle with the periphery is 90°, which reduces the cosine to zero and cancels the value of

$$-\frac{u\,V_2\cos a}{g}.$$

The vanes would therefore take the shape represented in Fig. 37.

Fourth. That a small fan at a high speed is equal to a large fan at a low speed, if the peripheral velocity in both is equal,

and the orifice of passage in the small fan is sufficient to keep the velocity of entry down within practical limits.

Fifth. According to the theoretical demonstration, the length of the chimney and the ratio of its expansion would only be correct for one quantity and velocity. The area of the discharge passage can be regulated, as in the Guibal fan, by a sliding shutter; but by lowering the shutter to suit any increase of fan speed and pressure, the corresponding increase of velocity will generate more friction and tend to counteract the effect of the lowered shutter. Another advantage derived from the cover and chimney is that it prevents the wind from affecting or baffling the current. The chimney, by discharging upwards, prevents the wind from obstructing the discharge. This is, perhaps, of as much practical importance as the gain in depression due to cover and chimney, as the pressure of the wind would often be an obstruction.

We also see from this theory that the pressure found in practice will fall far short of the initial depression H, because of imperfections, and also that the depression varies as the square of the speed, and of course the volume as the speed; to double a fan's speed gives nearly four times the water-gauge and double the air within the limits of reduction due to imperfections.

Ques. 2.—Explain the main features of construction you would suggest in the erection of a fan.

Ans.—At large and gaseous collieries, where dangerous volumes of gas have to be encountered, the fan should be a substantial fire-proof structure. If the whole structure is built of iron, some allowance might be required for expansion. By building the sides of brick, with the circular cover and stack of iron, a good result is obtained, as the brick wall can be easily removed for repairs.

The drift connecting a fan to a shaft should be fitted with a light movable cover, so that in the event of an explosion the rush of air up the shaft would lift it off and act as a safety-valve to prevent damage to the fan, with a duplicate cover ready to place quickly in position and restore the ventilation at once.

We would also prefer the fan to take air on both sides, and be fitted with doors, so that the current could be reversed in

case of accident from fire at the downcast, or to clear it from ice when necessary. The fan should be driven by a horizontal engine direct on the fan shaft, and have sufficient power to increase the speed one-half above the ordinary requirements of the mine.

For small mines, where the danger from gas or fire is not so great, then smaller fans built of iron with timber framing and cover would be sufficient.

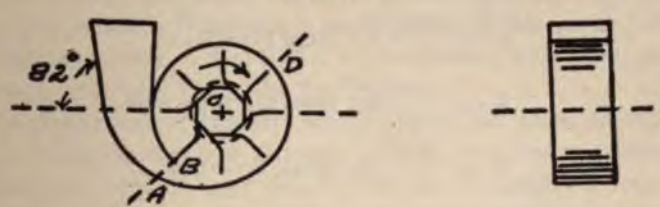
Although the theoretical depression of a 10-foot fan at 200 revolutions may be equal to that of a 20-foot fan at 100 revolutions, the passage through the small fan being less must have more resistance for the same volume; if the width was increased to partly make up the difference, still the speed of entry would be high, which would increase the value of

$$-\frac{V_0^2}{2g}.$$

A fan very wide in proportion to its diameter would have a longer distance for the current to travel in the direction of the axis, and at a high velocity the change of direction at right angles to follow the radius would produce more resistance. The diameter and width of a fan should, therefore, be in proportion to the work required.

The Editor uses the following formula for designing mine fans:

FIG. 38.



d = diam. of wheel in ft.
i = diam. of inlet in ft.
w = width of wheel in ft.
Q = quantity of air delivered in cubic ft. per minute.
r = radius of wheel = $\frac{1}{2}$ d.

r' = distance B C in ft.
r'' = distance A C in ft.
t = throat ordinate A B (in clear).
u = tangential velocity in ft. per second.
u' = tangential velocity in ft. per minute.
h = effective head in feet of air.
g = 32.2.

For large type fans	$d = \frac{1}{12}\sqrt{Q}$
For medium type fans	$d = \frac{1}{20}\sqrt{Q}$
For small type fans	$d = \frac{1}{33}\sqrt{Q}$

For plane-sided fans (when blades do not taper):

For large type fans	$w = \frac{1}{3}$ d.
	$i = \frac{4}{10}$ d.
For medium type fans	$w = \frac{4}{10}$ d.
	$i = \frac{1}{2}$ d.

If the blades taper or grow narrower from the inlet to the circumference then the greatest width should be greater than that given above.

For throat ordinate:

$$\log.\left(\frac{r''}{r'}\right) = \frac{.4343Q}{w\,r\,u'}$$

$r'' - r' = t$
$r' = r +$ clearance, casing, etc.

For speed:

$$h = \frac{.6\,u^2}{g} \text{ for large type fans.}$$

$$h = \frac{.5\,u^2}{g} \text{ for medium type fans.}$$

$$h = \frac{.3\,u^2}{g} \text{ for small type fans.}$$

It will be noticed that after the water gauge, quantity of air, and type of fan are decided, the dimensions can be all computed from the above. The Editor prefers the blades to begin with Murgue's angle of no shock at the inlet and end radially at the circumference. F. E. B.

At collieries where much gas is given off, the fan should have

a self-registering indicator both for speed and pressure, and be fitted with a governor.

Ques. 3.—Give your views as to the conditions to which each of the different types of fans is best adapted.

Ans.—The experiments which have been made to determine the efficiency of fans tend to the opinion that the Guibal has reached the highest percentage, some claiming even 65 to 70 per cent. of usual effect.

The Schiele fan has also given some good results, and is now extensively used.

The Guibal will give the best results where high water-gauge is required, and the Schiele will give the best results where the pressure required is small and the volume large in proportion to the dimensions of the fan. The Guibal fan has given good results at most of the deep collieries where the pressures required are from $1\frac{3}{4}$ to $2\frac{1}{2}$ inches of water-gauge. But where the ventilating pressure may be less, then a properly constructed Schiele would give better results and cost less. Experience in the anthracite region shows better results from a fan having the vanes made on the lines of the Guibal, with the cover fitting close to the periphery for a distance equal to the length of two or three spaces between the vanes, so that there will always be two vanes close to the cover, to prevent air from the centre at a lower pressure escaping round into the higher pressure where it is expelled into the chimney. From this point the cover gradually leaves the periphery, the space gradually widening, the expanding chimney being only a continuation of this spiral. Fans of this type are, therefore, a cross between a Guibal and a Schiele, and have given the best results.

In a steep-angled colliery, where the air has to be forced for long distances through airways and up and down manways, where the colliery may require 60 to 70 thousand feet per minute under a pressure of from $1\frac{3}{4}$ to 2 inches water-gauge, then the fan should be brought nearer the Guibal by having the spiral space to increase very little. In a colliery where the angle of dip may be less than 30°, where there are large openings, and the currents can be conveniently split, then there should be a fan nearer the Schiele, by making the spiral space

to increase rapidly and terminate in a short, wide chimney. Where the volume required is more than 100,000 cubic feet per minute, with from 1 to 1½ inches of water-gauge, this kind of fan will give the best results.

FIG. 39.

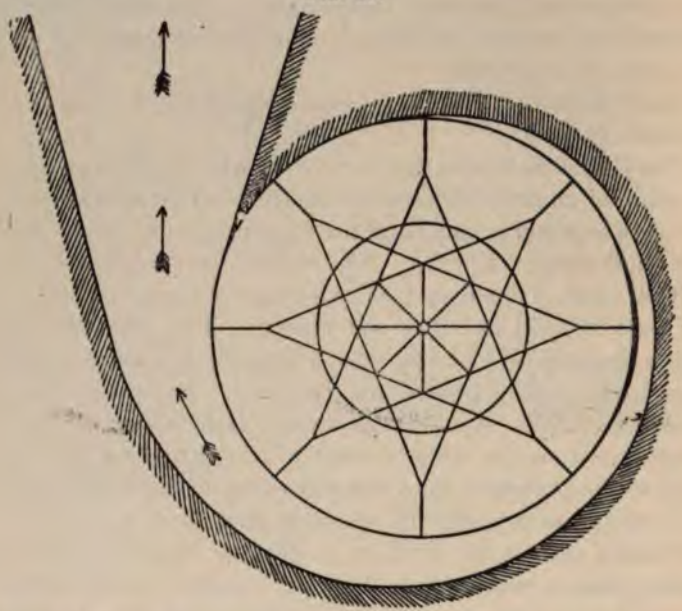


Fig. 39 represents the type of fan which we would consider best adapted to large volumes at moderate pressures. Where such fans have the spiral open or closed, as the mine may require, there is very little room for improvement. The plan of driving a fan with either a belt or rope is convenient for applying long-stroke engines which may be on hand, but directly connected engines are certainly less liable to accident from breaking of belts or bands, and where the engine is built expressly for the fan, direct action should be preferred.

FURNACES.

Ques. 4.—How would you construct a furnace to ventilate a mine where large quantities of fire-damp or black-damp are

given off, where the upcast shaft is, say 8 × 8 feet area and 500 feet deep, the mine employing about 150 persons under ground.

Ans.—The law requires 200 cubic feet per minute for each person employed. This would be 150 × 200 = 30,000 cubic feet per miuute as the least quantity allowed by law; and allowing for emergencies, say $\frac{1}{3}$ more, or 40,000 cubic feet as the full volume. This quantity can be passed through an average colliery, which would employ 150 hands, with a ventilating pressure of $\frac{8}{10}$ of an inch water-gauge.

Assuming water to be 833 times heavier than air, a water column of 0.8 would be equal to a column of air

$$0.8 \times 833 = 666.4, \text{ and } \frac{666.4}{12} = 55.5 \text{ feet}$$

in height. Air expands $\frac{1}{459}$th of its volume at 0° F. for every degree of heat added to its temperature, and the shaft is 500 feet deep, and 459 comes

$$500 \div 459 = 1.089$$

times in the depth of the shaft; therefore every degree of heat added to the upcast shaft will expand it 1.089 feet, and as it requires to be expanded 55.5 feet to produce the water-gauge, if we divide the required motive column by the length of expansion in the upcast for every degree, the result will show how many degrees the upcast will have to be raised in temperature:—

$$\frac{55.5}{1.089} = 50.9$$

degrees of heat to be added to the air by the furnace. If the air after passing through the furnace lost, say, 1° for every 10 feet of ascent in the upcast, it would lose

$$\frac{500}{10} = 50 \text{ degrees;}$$

therefore the furnace would need to impart to the air

$$50° + 25° = 75° \text{ of heat,}$$

to average 50° in the shaft.

The area of the upcast being $8 \times 8 = 64$ square feet, and the volume 40,000 cubic feet, the velocity in the upcast will be

$$\frac{40,000}{64} = 625 \text{ feet}$$

per minute as a final velocity, which is too great a speed for air in passing through a ventilating furnace. A high speed does not allow the air time to absorb heat, and is dangerous for carrying fire back to the upcast shaft, and is very inconvenient while cleaning the fire.

In this case we would prefer having two small furnaces instead of one large furnace.

Two furnaces can be fired alternately, so that one will help to burn the other's smoke. They also produce a more constant current, and when repairs are needed one can be kept in action to prevent accumulations of gas. Where top slate requires to be blasted down to make height for a furnace, two small arches will not require so much height as one large arch; therefore the difference in expense will be trifling. The next consideration is the location.

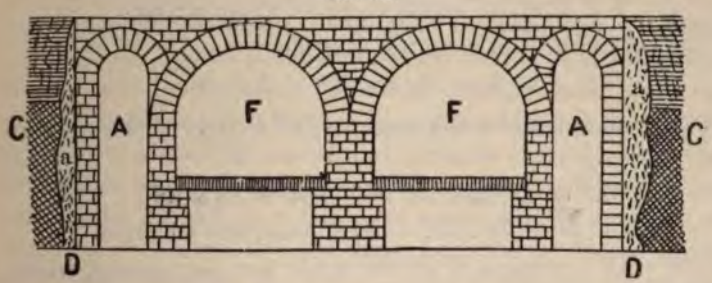
The heated air, after leaving the furnace, should have an ascending passage to the upcast shaft, and not be conducted along even a level, because immediately behind the furnace the air has the most heat and greatest buoyant power, which should be allowed to act in an ascending flue to start the velocity up the shaft. It is also necessary when rekindling the furnace, so as to carry off the smoke. Where the coal seam has considerable angle of dip and the furnace must be placed to the *rise* of the shaft, then the flue to the upcast would require to be driven as a graded tunnel in the measures over the coal and enter the shaft above the seam; but where it is sometimes necessary to remove accumulations of gas, a dumb-drift should be provided to carry the return air direct into the upcast without passing through the furnace. This dumb-drift should connect with the upcast at a distance sufficient to prevent flame from the furnace reaching the return air from the mine. To do this for both the furnace flue and the dumb-drift on the upper side of a shaft would be very expensive. By placing the furnace on one side

parallel with the strike of the measures and the dumb-drift on the opposite side of the shaft, and lining the flue with brick, the expense of the rock tunnels is avoided. Where a colliery has two shafts, one to the *rise* of the other, then the furnace can be located between them, and the airway from the furnace will have the angle of dip as ascent to the upcast, while the dumb drifts from each side can be driven from the return to the upcast in the seam.

The dumb-drift should have doors, so that the return can be passed through the furnace or dumb-drift at pleasure, the doors to lock in either position, so that none but those in charge can change them.

When the air is charged with gas, then the return should pass to the upcast by the dumb-drift, because passing gas through the furnace is dangerous; and when the return air is charged with black-damp it should be carried direct to the upcast by the dumb-drift, as passing it through the furnace would prevent the

FIG. 40.



fire from burning; but, under ordinary conditions, in a properly managed colliery the dumb-drift is only intended as a safety appliance for the removal of gas. The use of the dumb-drift weakens the effect of a furnace; but where the return air is charged with black-damp the furnace must be supplied with fresh air from the intake current.

The next considerations are the dimensions and construction of the two furnaces to fulfil the requirements of this question.

Fig. 40 represents a front elevation of two such furnaces placed in a seam 6 feet thick; the dimensions are as follows:

Width of furnace, 6′.
Length of bars, 6′.
Height to top of bars, 3′.
Spring of arch above bars, 2′.
Radius of arch, 3′.
Thickness of walls and brick arch, 1′.
Height over all, 9′.
Width of cold air spaces on each side, 2′.
Height of air spaces, 8′.

In building such a furnace a space 24 feet wide and over 9 feet high will be necessary. The space marked *a* on the figure should be filled with sand, carefully sieved to clean it of any combustible matter. The air spaces *A A* are to carry air currents between the furnaces and the coal *C*, and to carry off the heat, and are made 2 feet wide, so that a man can walk in on both sides to make inspections or repairs.

The areas of the two furnaces and air spaces are as follows:

Air spaces—

$$2 \times 8 = 16, \text{ and } 2 \times 16 = 32 \text{ square feet.}$$

Furnaces from bars to spring of arch—

$$2 \times 6 = 12, \text{ and } 12 \times 2 = 24.$$

Two semicircles of 3 feet radius equal a circle of 6 feet diameter.

$$6^2 = 36, \text{ and } 36 \times 0.7854 = 28.2744;$$
$$32 + 24 + 28.2744 = 84.2744 \text{ square feet area,}$$

which, with a volume of 49,000 cubic feet per minute, would give a velocity of

$$\frac{40{,}000}{84.27} = 474.66 \text{ feet}$$

per minute, a velocity high enough for air to pass a furnace.

The power of furnaces depends on so many different conditions that no fixed rules can be laid down as to their dimensions, but practically we consider the above as a size likely to comply with the terms laid down in the question. The flue to the shaft should have the sides lined with brick backed with sand rammed behind the wall wherever the sides are coal. The

ash-pit should have a few inches of water standing in it, and a hose from a water-pipe always ready for fire.

STEAM JET.

Ques. 5.—Describe the principle of the steam jet and its application as a mine ventilator.

Ans.—The steam jet when used to ventilate mines acts by imparting some of its momentum to the air. As it issues from the pipe at a high velocity it expands, and its increased surface coming in contact with the surrounding air imparts its momentum and thus produces motion.

The application of the steam jet to ventilate mines has not been very successful, and it is not calculated to deal with large volumes of air; the useful effect obtained for the coal consumed being small in comparison with that obtained by a fan or a furnace. We have seen it applied at the bottom of a shaft 300 feet deep, where 40 to 50 men were employed, and the current moved was sufficient to keep the workings in fair condition; but the seam was nearly horizontal, which allowed of large airways and small resistance. One great objection was the noise it made at the bottom of the shaft, interfering with conversation and even with the signals. This inconvenience could have been removed by placing it at such a distance from the top as to allow the steam to impart its momentum to the air and its heat to the sides of the shaft, and the steam to be condensed just before reaching the surface; but the noise would still be a source of annoyance at the top, and the water from the condensed steam falling down the shaft has a very injurious effect on the upcast current. If the jet is arranged to blow down, this objection is overcome; but in winter the cold air coming directly in contact with the steam condenses it and greatly reduces the effect.

The best result is obtained by the steam issuing from an annular orifice, thus increasing the surface in contact with the air. The jet should be inserted in the bottom of an expanding chimney like the discharge of a fan. By this arrangement the speed of air in passing through the contracted opening is brought much nearer the velocity of the steam, and the loss of steam momentum is less in overcoming the inertia of the air. Then

the current being slowed in the expanding chimney has the same effect as with the fan, the velocity being changed to pressure at the point of expulsion into the surrounding atmosphere. The steam jet, although not an economical ventilator, may often be used to advantage for temporary purposes.

Experiments with the steam jet have led to the following conclusions:

1. That the quantity of air put in motion by a steam jet depends upon the extent of surface in contact between the air and steam.

2. That the intensity of the vacuum increases in direct proportion to the steam pressure.

3. That the volume of air moved is in inverse proportion to the density of the air acted upon.

4. That the ventilating pressure obtained by forcing or exhausting is equal with equal steam pressure.

It is evident that the lower the temperature of the air against which the steam acts the greater will be the effect of condensation; therefore the efficiency will be greatest where the inside temperature is the highest and the jet placed in the upcast.

Where a steam jet is put in for temporary purposes it should be placed so that its *heat*, as well as its momentum, will act in inducing a current, and at such distance from the surface that the steam will have given out all its *heat* before leaving the upcast.

AIR MEASUREMENT.

Ques. 6.—Describe in detail how you proceed when measuring the air current in a mine.

Ans.—If measuring with an anemometer of the revolving type, select a place where the section is regular in outline, for convenience in finding the area, and free from eddies set up from narrowing, bending, or obstruction. Hold the instrument at arm's length and at right angles to the current; with a watch in the other hand time the revolutions. Take measurements over different parts of the area, because the velocity is generally highest in the central parts of the section if the current is not deflected by bends or angles in the airway. Add the several

measurements together, and divide by their number to obtain the average. This average velocity in feet per minute multiplied by the area in square feet gives the quantity passing in cubic feet per minute.

For accurate measurement there is a constant to be added to the indicated velocity, which represents the velocity due to the force necessary to overcome the friction of the instrument, the value of the constant being found for each anemometer by experiment. The space covered by the operator's body standing in the current should be deducted from the area, which for an average-sized man may be about 8 square feet.

For all practical purposes this and the constant may be neglected.

In using instruments without revolving vanes, such as "Dickinson's anemometer," the same mode of procedure is necessary, except that there is no watch required; it being only necessary to note the amplitude of the oscillations on the indicator and take the velocity midway on the scale between the limits of oscillation as the actual velocity.

The velocity of air-currents is sometimes ascertained by timing the speed of powder smoke, but this method is not adapted to high velocities. To measure with powder smoke, select a place where the passage has a very regular section for 50 to 150 feet, mark both ends and measure off in feet, take a small quantity of powder and explode it 10 or 12 feet to the windward of the first mark, and find the time the smoke takes to reach the other mark; then reduce the time and distance to feet per minute, by dividing the distance between the marks in feet by the number of seconds the smoke takes to travel that distance, and multiply by 60 to put it into feet per minute, and again by the average area between the marks to find the quantity in cubic feet per minute. This mode of measuring air is best adapted to velocities more than 100 and less than 500 feet feet per minute.

The air current passing in a gangway or tunnel with a velocity from 100 to 500 feet per minute can be measured correctly enough, for all practical purposes, with a watch and a common miner's lamp. Put a mark on the side, and pick up the cotton

until there is a large flame. With the lamp in one hand and the watch in the other, step off from the mark. Keep moving along with the current so that the flame of the lamp is not deflected. Keep moving for, say, 30 seconds and stop, making another mark. Measure the distance, repeat several times and take the average length, then the number of feet in 30 seconds doubled will be the feet per minute, and this again by the average area will be the quantity in cubic feet per minute.

When measuring to report in compliance with the law, always measure at the same place, and, if convenient, always during dinner-time or when the cars are not in motion, so that the current is not affected by the opening of the doors.

In measuring under a fan in a current of high velocity, a lantern is necessary to keep light, but we have often made measurements in the dark, with an assistant. First note the position of the pointers on the different dials of the anemometer, then get into a proper position by the light from the open door in the fan drift, close the door, and the assistant, with the watch, gives three distinct raps. At the third tap, start off the instrument. At the end of ten minutes the assistant again signals in the same manner to stop off the instrument; then subtract the first reading from the second, and the remainder is the velocity for 10 minutes, which, divided by 10, is the velocity per minute, and the area being known the quantity is found by multiplying the feet per minute by the area. Measure at different points of the area and take the average as before.

Ques. 7.—Suppose one side of the working from a shaft 300 yards deep to be filled with inflammable gas, and the danger attending its presence preventing both being worked; describe in detail what means you would adopt to expel the gas and restore the ventilation.

Ans.—Prohibit all fires or lights of any kind, except safety-lamps in or around the shaft. Start the fan and allow it time to clear the shaft and around the bottom, then descend the shaft, examine the air stoppings, and repair and stop all leaks in stoppings, doors, overcasts, etc. Place a canvas regulator across the intake to the side containing the gas, so as to regulate the amount in proportion to the return from the workings on

the clear side. Conduct the current into the first chamber filled with gas, by opening the cross-heading.. When the gas begins to move examine the return air with the safety-lamp. If the lamp shows gas, reduce the current from the foul side by means of the regulator. When the chamber is cleared open the next cross-heading and close the last, to clear the next, and so on, clearing out the gas in sections, and watching its effect on the return air, and regulating the supply from both sides so as to keep the main return up the shaft, below the explosive point. By thus removing the gas in sections, the ventilating pressure has more effect in overcoming the buoyant power of the gas, and the supply can be better controlled. The size of section to be cleared at one time will, to a great extent, depend on the angle at which the workings dip.

In steep workings the gas requires more force to drive it down to the returns, and must be taken in smaller sections. In level workings care must be taken that too much gas is not put in motion at one time, as it might cause danger in the returns. Of course, there is less danger with a fan than a furnace; still, by mixing the return from the gas with air from the clear side to make the return non-explosive, is the proper method.

But suppose you had no fan?

Then I would place a steam-jet in the upcast to produce a current and proceed in the same manner as with a fan, which, being much less powerful to move a current, would take a longer time.

But suppose you could not obtain enough steam to run a steam-jet of sufficient power to remove the gas?

Then I would use the steam-jet to clear the shaft and around the bottom, so as to get down to erect a furnace on the clear side, and by supplying the furnace with air from the side clear of gas, and allowing just as much to enter the upcast from the foul side as would make the return non-explosive, thus removing the gas in sections as with a fan, but at a slower rate and taking more time.

But suppose there was no steam available or material to construct a steam-jet, how would you get the shaft or bottom ventilated to erect a furnace?

Start the pumps and place an overflow in the pump landing box, so as to cause a heavy fall of water in the downcast to cause a temporary ventilation until a furnace could be erected.

But suppose the pumps were just able to keep the mine clear of water, and allowing water to fall down would overpower them?

Then I would erect a temporary furnace on top with a stack as high as practicable and connect it with the upcast, having a side door in the connection so that the furnace could be supplied from the surface or the upcast, and regulate the supply of fresh air to suit the amount of gas which might come from the mine.

But suppose you could not obtain the material to erect a stack on the surface?

Then I would connect the upcast to the boiler fires with a temporary structure to maintain sufficient ventilation at the bottom of the shaft to allow of a furnace being erected to clear out the gas.

CHAPTER VI.

MAPS—PILLARS.

Ques. 1.—If you were examining the map of the working in a colliery and measuring a place on the map driven straight from the gangway or entry, found its length to be $1\frac{3}{4}$ inches, how would you find the actual length of the place represented, the scale being 100 feet to one inch, and the place having an inclination of 50°?

Ans.—The map being 100 feet to the inch, $1\frac{3}{4} = 1.75$ inches, and $1.75 \times 100 = 175$ feet. This being the actual length of the place, reduced to horizontal measurement or the cosine of the angle, if the length as represented on the map is divided by the cosine of 50°, the result will be the actual distance or length of the place as measured on the slant in feet, or

$$\frac{1\frac{3}{4} = 1.75 \times 100 = 175}{\text{cosine of } 50^\circ = 0.64279} = 272.25 \text{ feet.}$$

If you were examining the maps of adjacent collieries, and it was necessary to find how much deeper a given point in one colliery was below a given point in another, how would you find the difference?

In all properly drawn maps of collieries the important points are marked, showing their elevation above or below the level of the sea at mean tide, this base line or level having been carried from the sea to the mines by fixed points and all maps calculated from that base. Therefore in order to find the difference in depth between any two or more points in different collieries we have only to compare from the tide elevations by either adding or subtracting from each other. This shows the difference between important points and the difference of intermediate points, aud these can be found by measurement of distance and dip, as marked on all properly constructed colliery maps.

Ques. 2.—If a chamber or working place at an angle of 30° with the gangway is driven 100 yards and it is desired to connect the end of said chamber with the gangway at the nearest point to establish ventilation, what distance would the shortest possible connection have to be driven; what distance along the gangway from the chamber would it have to be commenced, and at what angle would it leave the gangway and strike into the chamber?

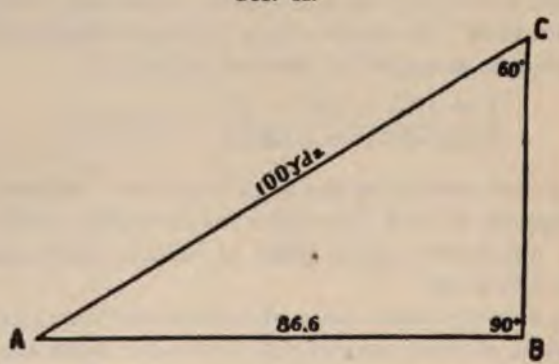
Ans.—Let *A C* (Fig. 41) be the chamber, *A B* the gangway, and *B C* the connection. Then because the angle *C A B* =30° and the sine of 30° = 0.5, the distance

$$C\,B = \tfrac{1}{2} \text{ of } A\,C, \text{ or } 0.5 \times 100 = 50 \text{ yards},$$

and it is evident that *C B* at right angles to the gangway *A B* is the shortest possible connection. Because the angle

$$C\,A\,B = 30°,\ 90° - 30° = 60°,$$

FIG. 41.



therefore the connection leaves the gangway at an angle of 90° and joins the chamber at an angle of 60°. The angle *C A B* = 30°, and the cosine of 30° = 0.86603, therefore the length of *A B* is

$$0.86603 \times 100 = 86.6 \text{ yards}$$

along the gangway from *A*.

Ques. 3.—Answer the same question in the case of the chamber making an angle of 45° with the gangway.

When the angle is 45° (Fig. 42) the chamber forms the diagonal of a square, and the results are as follows:—

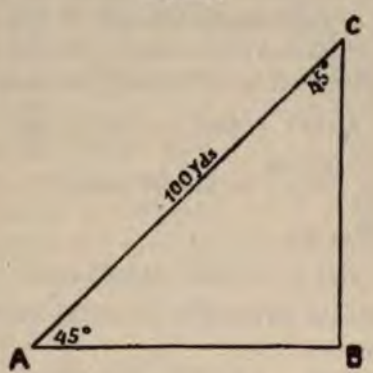
$$AC^2 = AB^2 + BC^2;$$

therefore

$$100^2 = 10{,}000, \frac{10{,}000}{2} = 5{,}000 \sqrt{5000} = 70.71 +;$$

therefore the lengths of AB and BC are each 70.71 yards, and the angle BCA is also 45°.

Fig. 42.



Ques. 4.—How would you lay off on the ground a square having sides 12 feet, having only a common tape line 12 feet long?

Ans.—Lay off a straight line with the tape in the position of a side of the required square from the point where you wish to place the corner of the square; measure off on the line a distance of 4 feet; take 8 feet on the tape and stick a pin through it at the 3 feet mark; hold the end of the tape at the corner end of the 4 feet line and the 8 feet mark at the other end; move the pin until both parts of the tape are tight, and mark the place, then the 3 feet part of the tape is at right angles with the 4 feet line. You will now have the corner of your square; measure 12 feet from the corner in both directions and you have two sides of the square; from the end of both lines, with a 12-foot radius, mark two segments, and their point of intersection will complete the square.

Ques. 5.—A colliery which has been worked down two lifts, on a steep north dip, is full of water. The slope is down 200 yards. From the mouth of the slope north a horizontal distance of 452 feet a shaft has been sunk cutting the seam at a depth of 840 feet. How far from the shaft level can the chambers be driven to leave a chain pillar under the slope gangway 40 feet thick, and what is the angle of dip?

Ans.—The length of chamber required by the conditions of this problem can be found either by construction or calculation. It is evident that the horizontal distance and the shaft are two known sides of a right-angled triangle of which the length of the slope + the thickness of the chain pillar + the length of the chambers from the shaft level is the hypothenuse; therefore

$$\sqrt{452^2 + 840^2} = 953.88 \text{ feet},$$

$$\frac{953.88}{3} = 317.96 \text{ yards};$$

the slope and pillar are

$$200 + 13.33 = 213.33 \text{ yards},$$

which taken from the full length leaves the length the chambers can be driven up,

$$317.96 - 213.33 = 104.63 \text{ yards}.$$

If we now divide the depth of the shaft by 953.88, the result will be the sine of the angle of dip, or dividing the horizontal distance by 953.88 will give the cosine.

$$\frac{452}{953.88} = 0.47384$$

as the cosine of the angle of dip. By examining a table of sines we find that this is more than 61° and less than 62° of dip.

Ques. 6.—A seam of coal crops out on the south side of a valley, and dips north at an angle of 75°; the side of the hill has an inclination of 40°; from the outcrop of the coal down the hill 100 yards a shaft is being sunk; if the coal has a uniform dip, how deep will the shaft be when it reaches the coal?

Ans.—This question can be best explained by a figure. Let *AB* (Fig. 43) represent the profile of the hill at an angle of 40°, *AD* the coal seam at a dip of 75° north, *A* the point of

outcrop, and B the top of the shaft 100 yards down the side of the hill. Let C be the imaginary point vertical over the shaft, then the lines CB and CA will be the sine and cosine of the triangle ABC respectively; therefore, their lengths will be found by multiplying the sine and cosine of 40° by 100.

Fig. 43.

The sine of 40° = 0.64279.

The cosine of 40° = 0.76604.

$0.64279 \times 100 = 64.279$ yards $= CB$

$0.76604 \times 100 = 76.604$ yards $= AC$

Again, because the triangle ADC has the angle at $A = 75°$, and the line AC is the cosine; if we divide AC by the cosine, it will give the length AD, and the length of AD multiplied by the sine of 75° will give the length of CD.

The sine of 75° = 0.96593, and the cosine of 75° = 0.25882. Then

$$\frac{76.604}{0.25882} = 296.5 \text{ yards} = AD,$$

and

$$296.5 \times 0.96593 = 286.5 = CD.$$

If we now take CB from CD, it will leave BD

$$286.5 - 64.279 = 222.221 \text{ yards};$$

$$222.221 \times 3 = 666.7 \text{ feet, or}$$

$$\frac{0.96593 \times 76.604}{0.25882} = 286.5$$

$$286.5 - 64.279 = 222.221 \times 3 = 666.7 \text{ feet}$$

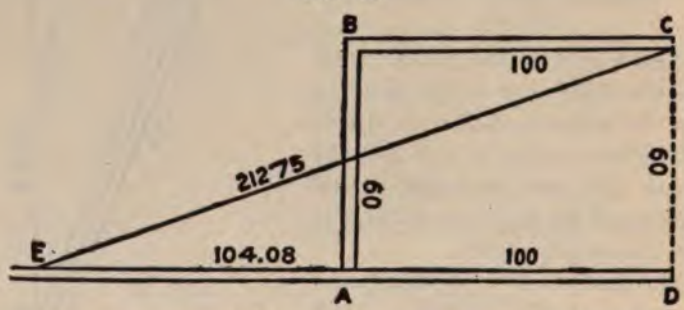
as the depth of the shaft. This problem can also be solved by construction.

Ques. 7.—On the gangway of a mine a counter chute is driven

up 60 yards, the angle of dip being 18°. From the top of the counter chute east a gangway has been driven 100 yards to prove the coal. The counter chute having too little grade to cause the coal to slide to the gangway on sheet iron, it has been decided to reach the face of the counter gangway by a slant road driven at an inclination of 5°, so that the cars can be run down to the gangway with sprags. How far west from the bottom of the counter chute must the slant road be commenced to give the road the proper grade?

Ans.—This question can perhaps be best solved by the aid of a diagram. Let *AB* (Fig. 44) represent the counter chute up

FIG. 44.



60 yards at an angle of 18°. The length multiplied by the sine of 18° will give the vertical height of *B* above the gangway, which is the same height as *C*. The sine of 18° is 0.30902 and

$$60 \times 0.30902 = 18.5412 \text{ yards}$$

as the vertical height to be gained by the slant road when it reaches *C*. If the vertical height is divided by the sine of 5°, it will give the hypothenuse, or the required length for the slant road.

$$\frac{\text{vertical height } 18.5412}{\text{sine of } 5° = 0.08716} = 212.72 \text{ yards},$$

the length from *E* to *C*. The line of *CD* being equal to the chute is 60 yards, therefore the two sides *EC* and *CD* of the right-angled triangle *ECD* are known. If from the square of the hypothenuse *EC* is taken the square *CD*, the remainder will be the square of *ED*.

$$212.72^2 = 45{,}249.7984, \text{ and } 60^2 = 3600;$$
$$45{,}249.7984 - 3600 = 41{,}649.7984$$

the square of *ED*, the square root of which is its length

$$\sqrt{41{,}649.7984} = 204.08 \text{ yards.}$$

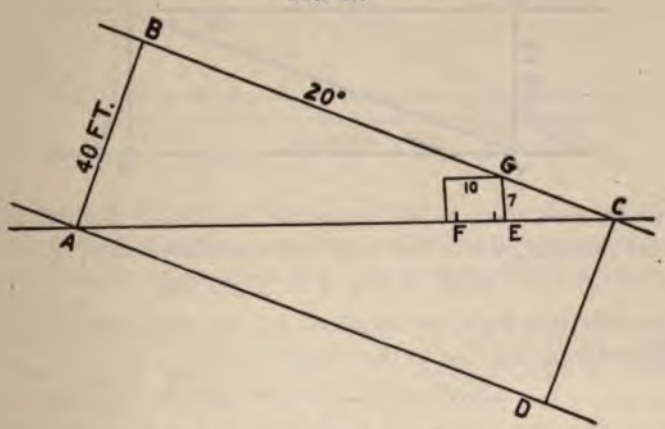
Again, because *AD* is equal to *BC*, its length is 100 yards, which taken from *ED* will leave *EA*.

$$204.08 - 100 = 104.08 \text{ yards,}$$

the distance west on the gangway from the bottom of the counter chute where the slant road will have to branch off from the gangway. This question can be solved by construction.

Ques. 8.—In a colliery where the seam is 40 feet thick on a dip of 20°, a gangway is driven against the top slate, the slate is not broken, and the gangway is 7 feet high and 10 feet wide. It is decided to drive a branch gangway to the bottom slate. By putting in a 20° frog, and running straight through it, how far will the branch have to be driven to strike the bottom slate, the gauge of the road being 4 feet?

FIG. 45.



Ans.—For the purpose of illustrating the demonstration of this proposition, let *ABCD* (Fig. 45) be a profile of the seam dipping at an angle of 20°; *AB* the thickness of the seam, equals 40 feet; *EG*, the height of the gangway, equals 7 feet; and *CA*,

a horizontal line from top to bottom of the seam, on the gangway level. Let us first determine the length of AC.

Because the triangle ABC has the angle at B a right angle, the line AB is the sine of the angle ACB; therefore if we divide AB by the sine of 20°, it will give the length of the hypothenuse AC, the sine of 20° is 0.34202,

$$\frac{40}{0.34202} = 116.95 \text{ feet} = AC.$$

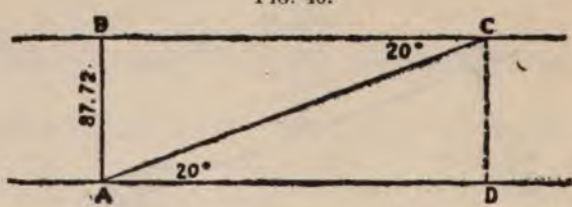
Let us next determine the length of CE. Because the triangle GEC has the angle at E a right angle, GE and EC are the sine and cosine of the angle ECG, respectively; the length of the sine being known, then, by proportion,

sine of 20° = 0.34202 : cosine of 20° = 0.93969 :: GE : EC, or

$$\frac{0.93969 \times 7}{0.34202} = 19.23 \text{ feet} = EC.$$

The gangway is 10 feet wide and the track 4, leaving 10 — 4 = 6 feet clear of the track. Of these 6 feet, 4 are on the lower

FIG. 46.



or drain side, next to E; therefore the length from C to the upper side-rail, where the frog has to be placed at F, is

$$19.23 + 4 + 4 = 27.23 \text{ feet.}$$

If we take this from the length of AC, the remainder will be the distance from the frog to the bottom slate,

$$116.95 - 27.23 = 89.72 \text{ feet;}$$

but as the upper side-rail has to stop 2 feet from the bottom slate, this is reduced 2 feet, leaving a distance from the frog of 87.72 feet. We have now found one side and an angle of the parallelogram, of which the branch gangway upper side-rail is the diagonal.

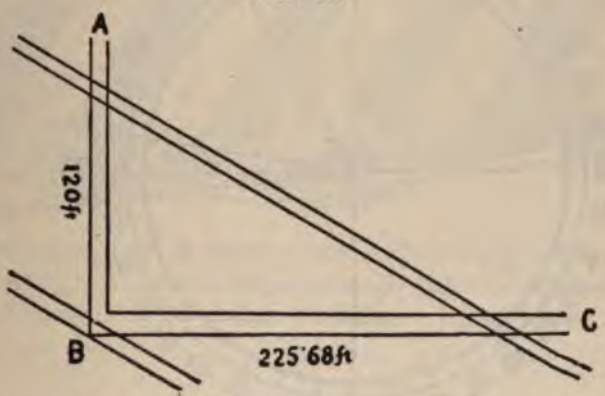
Let AB (Fig. 46) be the distance from the frog to bottom slate — 2 feet. Because the lines BC and AD are parallel, and the angle $DAC = 20°$, the alternate angle ACB is also 20°; and because the angle at B is a right angle, BA is the sine of the angle at C; therefore if we divide BA by the sine of 20° it will give the length of the hypothenuse AC, or

$$\frac{87.72}{0.34202} = 256.47 \text{ feet} = AC,$$

as the distance from the point of the frog to within 2 feet of the bottom slate.

Ques. 9.—A colliery is working a seam of coal having a dip of 28°. A small overlying seam was cut in the shaft 120 feet

FIG. 47.



from the bottom. What length of tunnel will be required to cut the overlying seam, and, when cut, what distance will have to be driven to connect with the upcast and establish permanent ventilation?

Ans.—In the diagram let AB (Fig. 47) be the shaft, crossed 120 feet up, by the coal seam AC. Because ABC is a right-angled triangle of which the perpendicular side AB is 120, then

sine 28° = 0.46947 : cosine 28° = 0.88295 :: 120 : BC, or

$$\frac{0.88295 \times 120}{0.46947} = 225.68 \text{ feet, or}$$

$$\frac{225.68}{3} = 75.22 \text{ yards,}$$

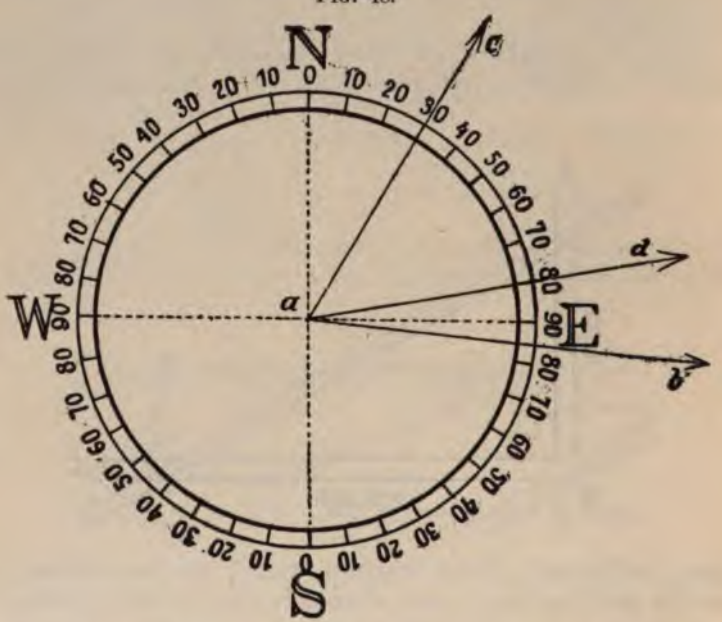
as the length of tunnel required to cut the seam AC. Again, the perpendicular AB divided by the sine of 28° will give the length of AC.

$$\frac{120}{0.46947} = 255 \text{ feet, or}$$

$$\frac{255}{3} = 85 \text{ yards,}$$

the distance required to connect, for ventilation.

FIG. 48.



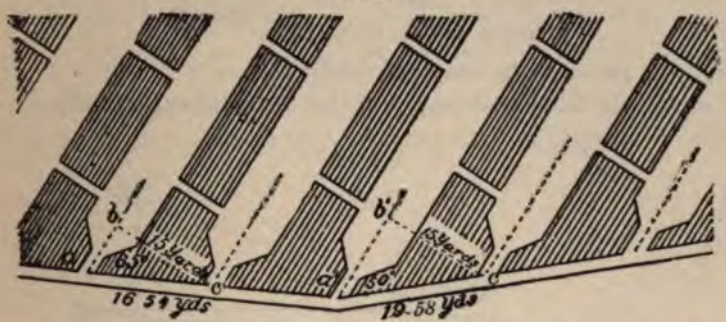
Ques. 10.—If an entry is driven S. 85° E. and the chambers N. 30° E., what should be the distance between the centre lines of the chambers measured along the entry, the chambers and pillars being 8 and 7 yards wide respectively? And if the entry is changed in direction to N. 80° E., what would be the difference in the distance along the entry to maintain the same width of chamber and pillar?

Ans.—The first thing to be determined is the contained angle between the direction of the entry and chambers. In Fig. 48,

let AB represent the entry bearing S. 85° E., AC the centre line of the chambers bearing N. 30° E. Then $90° - 30° = 60°$ in the NE. quadrant, and $90° - 85° = 5°$ in the SE. quadrant; and $60° + 5° = 65°$, the value of the contained angle BAC. Again, let AD be the bearing of the entry changed to N. 80° E., then $80° - 30° = 50°$, as the contained angle DAC.

A chamber and pillar are $8 + 7 = 15$ yards wide, the distance between the centre lines as measured at right angles to their direction. In Fig. 49, let AB be the centre line and BC

FIG. 49.



the distance = 15 yards between each. Then we have the right-angled triangle ABC, with the given angle $BAC = 65°$, and the side BC opposite = 15 yards. Therefore, by proportion,

sine of $65° = 0.90631 : 1 :: 15$ yards : hypothenuse AC, or

$$\frac{15}{0.90631} = 16.55$$

yards as the length of AC, the distance between the centre lines as measured along the entry.

Where the changed bearing of the entry reduces the contained angle $B'A'C'$ to 50°, then

$$\frac{15}{\text{sine } 50° = 0.76604} = 19.58$$

yards will be the distance between the centre lines measured along the entry, and

$$19.58 - 16.55 = 3.03$$

yards will be the difference caused by the change.

Rule.—Divide the sum of the width of chamber and pillar by the sine of the angle between their bearings.

Ques. 11.—If a map shows a distance of 200 feet between the entry and a land line parallel to it, how far can a chamber be driven without crossing the line, the dip being 48° and the entry driven level?

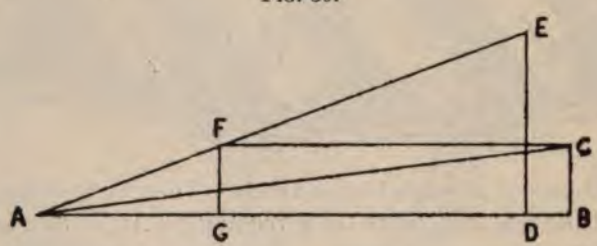
Ans.—Because the 200 feet represented on the map is the horizontal side of a right-angled triangle, of which the distance to be found is the hypothenuse and the dip the contained angle, therefore

$$\frac{200}{\text{cosine } 48° = 0.66913} = 298.89$$

feet, the distance the chamber can be driven.

Rule.—Divide distance on map by cosine of angle of dip.

FIG. 50.



Ques. 12.—On a seam of coal dipping 15° the chambers are driven slant across the dip, so that the car track will have a grade of not more than 8°. The chambers are 7 yards wide and the pillars 5 yards thick, what is the distance in feet between the centre lines of the chambers as measured along the entry?

Ans.—The sine of 15° is 0.25882; therefore at 100 feet up, at right angles to the entry, the vertical height gained would be 25 feet and 10 inches nearly.

The sine of 8° is 0.13917; therefore at 100 feet up the chamber track the vertical height gained would be 14 feet nearly.

From this point on the chamber track and at right angles to the entry, the distance down through to the entry is proportional to the sine of the contained angle between the entry and room track.

In Fig. 50 let AB and BC be the cosine and sine of 8° respectively, and AD and DE the cosine and sine of 15°. The height FG being equal to the height CB, gained at one foot up the chamber track; therefore the distance AF is the sine of the contained angle between the entry and chamber. Because by the principle of similar triangles

$$ED : FG :: EA : FA;$$

therefore by substitution we have the following proportion to find FA:—

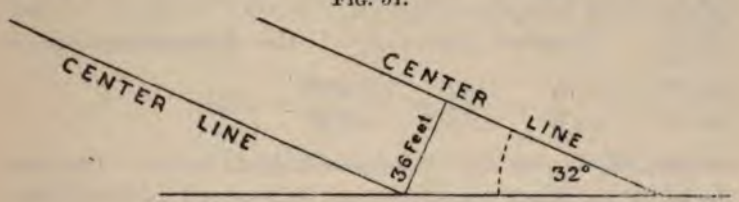
$$\text{sine } 15° : \text{sine } 8° :: 1 : FA, \text{ or}$$

$$\frac{0.13917}{0.25882} = 0.5378,$$

the sine of the contained angle nearly.

From a table we find this to be $32\frac{1}{2}$ degrees nearly.

Fig. 51.



The chamber and pillar are $7 + 5 = 12$ yards, or 36 feet wide, between the centre lines; therefore 36 feet is the side of the triangle opposite the contained angle, and is proportional to its sine, as shown in Fig. 51. The distance on the chamber track would be proportional to its cosine, while the distance along the entry would represent the hypothenuse, which is found by proportion as follows:—

$$\text{sine } 32° : 1 :: 36 : \text{distance along entry;}$$

$$\text{or } \frac{36}{0.5378} = 66.93,$$

or 67 feet nearly as the distance in feet along the entry between the centre lines of the chambers.

Rule.—Divide the sine of the angle of grade on the chamber track by the sine of the angle of dip to find the sine of the con-

tained angle between the entry and chamber. The width of chamber and pillar divided by this sine gives the distance along the entry between the centre lines of the chambers.

In applying this rule to find the compass bearings of rooms with a given grade it must be remembered that the angle so found is the angle measured on the coal; while the angles measured by the compass are on an imaginary level plane. As the entry in the above example is level, suppose we wish to find the bearings of the chambers when the bearing of the entry is S 40° W. To do this we must find the horizontal angle between the chambers and the entry. The sine of this angle is the shortest horizontal distance from the entry, of a point which is one foot from the entry when measured along the chamber on the coal; divided by the horizontal distance of that same point from the entry, when measured along the room. Or it is $AG \div 1 \times \cos 8°$. But $AG = FA \times \cos 15°$ and $FA = \frac{\sin 8°}{\sin 15°}$. Therefore the sine of the horizontal angle $= \frac{\sin 8° \times \cos 15°}{\sin 15° \times \cos 8°} = \frac{.13917 \times .96593}{.25882 \times .99027} = .525.$

And .525 = sine $31\frac{3}{4}°$ instead of $32\frac{1}{2}°$ as before. The bearing of the chambers would be S 40° W + $31\frac{3}{4}°$ = S $71\frac{3}{4}°$ W.

PILLARS.

Ques. 13.—Explain how you would determine the sizes of pillars in mines with seams of different thicknesses and at different depths from the surface.

Ans.—To go into all the minutiæ involved in this question would require a treatise on the working and winning of coal.

With the exception of ventilation, this is, perhaps, the most important subject connected with mining; the whole object in leaving pillars being to support the roof and prevent danger or damage until the time comes for their removal. It is imperative that we should have some idea of the relative resistance that pillars of different dimensions will have to the compression of the strata above them. If the pillars are too small, *squeezes* and *creeps* will be induced, causing danger from sudden and

extensive cave-ins and great expense for cribbing and timber, often resulting in districts of the mine being closed and large quantities of coal lost. If the bottom is soft, small pillars are apt to be forced down, causing the bottom between them to heave up. If the bottom is hard and the top brittle, there will be trouble and expense from falls; if both top and bottom are strong, then the coal will be crushed and the value of the pillars depreciated.

There are so many different conditions to be considered that it is impossible to lay down any rigidly correct rule for determining the size of the pillars, or, rather, the relation between the coal mined and the coal which should be left solid in proportion to the thickness of the seam and its depth from the surface.

The pressure upon the pillars of a mine is enormous. Assuming the average weight of the strata at 180 pounds per cubic foot, at a depth of 500 feet, the weight per square foot of surface on the coal would be

$$\frac{180 \times 500}{2000} = 45$$

tons; and when we have taken out 50 per cent. of the coal, the whole weight, being thrown on the other half, doubles the weight per unit of surface, making it 90 tons to the square foot. This is a pressure of

$$\frac{90 \times 2000}{144} = 1250$$

pounds per square inch, equal to

$$\frac{1250}{75} = 16.6$$

more than sixteen and a half times the working pressure on our boilers at the mines.

It is evident that, all other things being equal, the depth from the surface is a measure of the pressure; at double depth there will be double the weight. It is also evident that the amount of strata over a given area of coal will be in proportion, not to the area, but to the cosine of the angle at which the coal

lies. Where the seam is level, a pillar 10 yards square will have

$$10 \times 10 = 100$$

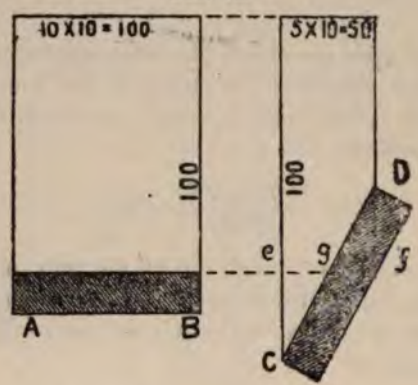
square yards of strata over it; but if it dips at an angle of 60°, because the cosine of 60° is 0.5, the horizontal measurement over the pillar will be only

$$0.5 \times 10 \times 10 = 50$$

square yards.

In the following example let AB and CD, Fig. 52, be two pillars 10 yards square and 100 feet below the surface, AB being

FIG. 52.



level and CD at a dip of 60°. Assuming the weight of the strata at 175 pounds per cubic foot, the weight of the strata above every foot of the pillar AB would be

$$175 \times 100 = 17{,}500$$

pounds; the pillar being

$$30 \times 30 = 900$$

square feet, the total would be

$$17{,}500 \times 900 = 15{,}750{,}000 \text{ pounds;}$$
$$15{,}750{,}000 \div 2000 = 7875 \text{ tons.}$$

The pillar CD is also 900 square feet, but at a dip of 60° the

horizontal line EF is only $\frac{1}{2}$ of AB, or 15 feet; therefore the total amount of strata and weight is $\frac{1}{2}$, or 3937$\frac{1}{2}$ tons.

The triangular piece of strata geC equals the triangular piece qfD occupied by the coal; therefore the loss of weight above the level of the average pressure at G is made up by the gain of weight below it. The term pressure used in this proposition must not be confounded with pressure as used in reference to water, air, or steam.

We therefore infer that the pressure or weight upon a pillar in a mine varies directly as the depth from the surface multiplied by the cosine of the angle of dip.

Of course, this theory is on the assumption that the mass of strata above a given area of coal receives no support in virtue of its strength or adhesion to the surrounding strata, which practically is only true over large areas.

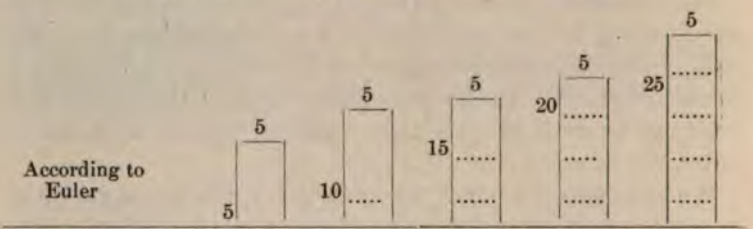
If the pillars in a mine are to be considered as columns supporting weight, then we would suppose that their relative strength in proportion to their dimensions will have to be determined by the laws which have been deduced from experiments made to determine the "strength of materials" by such investigatiors as Euler, Musschenbrock, Hodgkinson, Eytelwein, Fairbairn, and others.

According to the theory of Euler, the strength of a column of any solid material of prismatic form to resist crushing from weight placed upon it varies as the square of the number expressing its sectional area divided by the square of the number expressing its height. According to the experiments of Eytelwein, the strength of rectangular columns to resist compression in the direction of their height is directly as the product of their longest side multiplied by the cube of their shortest side and inversely as the square of their height.

These two theories coincide only when the pillars are square; therefore, according to both, the strength of square pillars will vary *directly* as the fourth power of their sides and *inversely* as the square of their heights.

The pillars in a mine are generally rectangular; let us, therefore, examine the result of both theories—first, when applied to pillars of equal breadth and different lengths; secondly, to pillars of equal length but different breadths.

Let the following diagram represent pillars and the figures in the column opposite each pillar the relative results.

According to Euler					
Area =	25	50	75	100	125
Area^2 =	625	2500	5625	10,000	15,625
Relative strength	1	4	9	16	25
According to Eytelwein	$5^3 \times 5 =$ 625	$5^3 \times 10 =$ 1250	$5^3 \times 15 =$ 1875	$5^3 \times 20 =$ 2500	$5^3 \times 25 =$ 3125
Relative strength	1	2	3	4	5

With pillars of equal breadth the first theory shows them to vary in strength as the square of their length, while the second shows them to vary only as their length, other things being equal. We are inclined to the opinion that both these theories are extreme. The second, third, fourth and fifth pillars are just so many pillars like the first placed together; and we know that three placed together solid will be stronger than if separate, because it changes the proportion between the section and the height; but that five pillars placed in a continuous wall are 25 times stronger than one seems rather too great a proportion of increase in strength. In the present state of our knowledge we would incline to follow Eytelwein and consider long pillars to increase only as their length; for although we are of opinion that they increase more than in that ratio, it leaves a margin for safety, and as the error will be on the safe side, we would prefer it rather than assume the strength of long pillars to vary as the squares of their lengths.

Let us now see the result with pillars of equal length but dif-

ferent breadths and placed on equal spaces, so that there will be equal weight to support.

According to Euler

Width	1	2	3	4	5
Area =	10	20	30	40	50
Area^2 =	100	400	900	1600	2500
Relativ strength	1	4	9	16	25
According to Eytelwein					
Breadth^3 × Length	10	80	270	640	1250
Relative strength	1	8	27	64	125

With pillars of equal length the first theory shows their strength to vary only as the square of the breadth, while the other shows them to vary as the cube of the breadth. In the first case, to double the width only makes the strength four times, while in the second case it makes it eight times greater. Both cannot be correct, and neither may be. In this case, as before, we would incline to follow Eytelwein, because we know that the resistance to transverse strain in beams or girders varies as the square of the depth, and we think the ratio for compression in the direction of the axis is greater. We therefore incline to the second theory and assume that to double the breadth of a pillar makes it not only 4, but 8 times as strong, so long as the pressure does not exceed the limits of resistance in the material.

The experiments all appear to agree as to the effect of height, and as the height of pillars is measured by the thickness of the seam, the height becomes an important factor. The following

diagram (Fig. 53) will show the relative strengths of pillars of equal area in seams of different thickness.

FIG. 53.

Height²	4	16	36	64
Relation between height²	1	4	9	16
Relative strength	1	$\frac{1}{4}$	$\frac{1}{9}$	$\frac{1}{16}$

These results lead us to the following conclusions in regard to the relative strengths of pillars in a mine, all other conditions remaining the same:—

1. That the weight per unit of surface varies as the depth from the surface over large areas.

2. That the weight on a given dimension in the direction of the dip varies as the cosine of the angle.

3. That the strength varies inversely as the square of the thickness of the seam.

4. That with square pillars, the strength varies as the square of the area of section, or the fourth power of their side.

5. That with pillars of equal length the strength varies as the cube of their breadth.

6. That with pillars of equal breadth, their strength varies as their length; when the length exceeds twice the breadth, and when the length is under twice the breadth, the strength varies as the square of the length.*

*The present editor does not agree with the author in these conclusions about the relative strength of pillars. It is a well-known fact that when the height of a pillar becomes less than about one-fourth of the least dimension,

Although these laws are the result of experiments made to determine the relation between dimensions and strength, they can only be approximate, and only apply where the conditions are equal, and conditions are unequal in all mines. But in the absence of anything better, we would determine the sizes of pillars from the above laws, or some slight modification of them.

Ques. 14.—If the pillars in a mine are 10 yards square at a depth of 300 feet below the surface, how many yards square should they be at a depth of 600 feet?

Ans.—If the strength of square pillars varies in proportion to the fourth power of their sides, and the strength required is in proportion to the depth, then as the one depth is to the other depth so is the fourth power of the side of the given pillar to the fourth power of the side of the pillar required, and the fourth root will, of course, be the length of the side.

$$1 : 2 :: 10^4 : 20{,}000,$$

and the square root of the square root will be the fourth root or side required;

$$\sqrt{20{,}000} = 141.42 +, \text{ and } \sqrt{141.42} = 11.93$$

yards; so to double the strength of a pillar 10 yards square would require it to be made 12 yards square.

If the strength varies as the fourth power of the side, then the side will vary as the fourth root of the strength, and as the

then it yields simply by crushing. The ultimate crushing load becomes independent of its height and is proportioned only to its area.

For chamber pillars we would suggest that no pillar be made of less width than the height of the seam and that the areas be proportioned to the load when all other conditions are equal. The roof, floor, and coal itself should be carefully studied and allowance made for structure, slips, etc. These pillars should be amply large especially if intended for subsequent removal.

Barrier pillars, pillars to protect haulage roads, shaft pillars and all pillars where it is important that the adjacent strata should not be disturbed, or that any disturbance should be checked, should be much larger. As the extent of disturbance is increased with the thickness of the seam, we think larger pillars of this class should be used in thicker seams. The pressure of water should also be allowed for. There is much difference of opinion among writers as to the proper method of determining the size of pillars. *Practical experience with the local conditions existing in the region wherein the new pillars are to be laid off seems to be the best guide.* F. E. B.

relative depths are a measure of the relative strengths required, then the sides will be in proportion to the fourth root of the depths, or

$$\sqrt[4]{1} : \sqrt[4]{2} :: 10 :: 11.93$$

yards, or the same result.

Ques. 15.—If the pillars in a mine are 12 yards square near the bottom of a shaft 1296 feet deep, how many yards square should they be near the outcrop, where the seam is 256 feet from the surface?

Ans.—

$$\sqrt[4]{1296} : \sqrt[4]{256} :: 12 : x =$$
$$6 : 4 :: 12 : 8$$

yards square, all other things being equal.

Ques. 16.—If the pillars in a mine are 8 yards square and separated in both directions by chambers 4 yards wide, the seam being level, at a depth of 300 feet below the surface, what length should the sides of square pillars be at a depth of 850 feet on a dip of 30°, the width of the chambers and all other things remaining the same?

Ans.—

$$300 : 850 :: 8^4 : 11{,}605.33, \quad \sqrt[4]{11{,}605.33} = 10.35 \text{ yards},$$

the pillar requiring to be $10\frac{1}{3}$ yards square. On the level every pillar has to support $(8 + 4)^2 = 144$ square yards of strata. On a dip of 30° the length on the dip would be as much longer than 12 yards as the hypothenuse is longer than the cosine; therefore, we will have to divide the 12 yards by the cosine to find the length of the dip:—

$$12 \div 0.86603 = 13.85.$$

This would be a rectangle on the dip of

$$13.85 \times 12 = 166.2$$

square yards of coal under each vertical 144 square yards of strata; this thrown into a square would have a side equal to

$$\sqrt{166.20} = 12.89 \text{ yards}.$$

Out of this we must take the enlarged pillars to find the width left for chambers at the increased depth—

$$12.89 - 10.35 = 2.54 \text{ yards}$$

left for chambers. But the chambers are to be the same as before; therefore the side of the square must be increased to

$$10.35 + 4 = 14.35.$$

This would be an area on the slant of

$$14.35^2 = 205.92$$

square yards.

We must now reduce this back to horizontal measurement, so as to find the proportionate weight over the increased space required for the larger pillars. 14 35 being the side of the increased square, multiplying it by the cosine will reduce it to the horizontal measurement.

$$14.35 \times 0.86603 = 12.42.$$

This changes the square of 14.35 on the dip to a rectangle of

$$14.35 \times 12.42 = 178.22$$

on the level, which is the strata over the increased pillars and chambers. Now the pillars were increased from 8 to 10.35 yards, to support 144 square yards at 850 feet from the surface; but to make space for chambers they have to support 178.22 square yards of strata; therefore the pillars will have to be still further enlarged to support the increased amount of strata over them.

$$144 : 178.22 :: 10.35^4 : 14{,}202.1909$$

$$\sqrt[4]{14{,}202.1909} = 10.91.$$

The pillars would, therefore, require to be 10.91, or nearly 11 yards square. There will always be a fractional difference because we have to calculate for a quantity which does not include the extra space required for the second increase of the pillars.

Ques. 17.—If pillars are 10 yards square in a seam 6 feet thick, how many yards square should they be in a seam 12 feet thick?

Ans.—If the strength of pillars varies inversely as the square of the thickness of the seam, then to make them stronger in a

...the following proportion: As the square of ... is to the square of the thickness ... so is the fourth power of the side of the ... to the fourth power of the side of the ..., and the fourth root will, of course, be ... of the side.

$$12^2 : 12^2 :: 10^4 : 40{,}000 =$$
$$12^2 : 2^4 :: 10^4 : 40{,}000$$

... of the square root will be the fourth root or

$$\sqrt{40{,}000} = 200, \quad \sqrt{200} = 14.14 +$$

... the length of the side, or 14 yards square.

... pillars in a seam 12 feet thick are 10 yards ... yards square should they be in a seam 4 feet ...

... 12-foot seam being 3 times as high,

$$3^4 : 1^4 :: 10^4 : 1111.11,$$

$$\sqrt{1111.11} = 33.33 \text{ and } \sqrt{33.33} = 5.77,$$

... yards square.

... If the pillars in a seam 6 feet thick and 300 f... ... the pillars are 10 yards square, what should be the size ... pillars in a seam 12 feet thick 900 feet below the surfa...?

$$1^6 \times 1 : 2^6 \times 3 :: 10^6 : 120{,}000.$$

$$\sqrt{120{,}000} = 346.41 \text{ and } \sqrt{346.41} = 18.61 \text{ yards square.}$$

... If square pillars in a seam 18 feet thick meas... yards along the sides, being 200 feet below the surface, w... ... should the sides of square pillars be in an underly... ... 300 feet below it and 6 feet thick?

$$1^6 : 12^6 \times$$

... yds.

Ans.—If the strength of pillars varies as the cube of their breadth, then

$$300 : 900 :: 8^3 : x =$$

$$1 : 3 :: 512 : 1536, \sqrt[3]{1536} = 11.5 \text{ yards.}$$

Ques. 22.—If pillars in a seam 8 feet thick are 7 yards broad at a depth of 250 feet, how broad should the pillars be in a seam 20 feet thick at a depth of 875 feet?

Ans.—

$$250 \times 8^2 : 875 \times 20^2 :: 7^3 : x =$$

$$16 : 350 :: 343 : 7503.125$$

$$\sqrt[3]{7503.125} = 19.5 \text{ yards broad.}$$

Ques. 23.—If pillars in a seam 30 feet thick are 12 yards broad at a depth of 343 feet, what breadth should pillars be at a depth of 1000 feet in a seam 15 feet thick?

Ans.—

$$343 \times 2^2 : 1000 \times 1^2 :: 12^3 : 1259.475.$$

$$\sqrt[3]{1259.475} = 10.7 + \text{ yards broad.}$$

Ques. 24.—In a seam of coal 8 feet thick at a depth of 500 feet below the surface, if the chambers and pillars are 9 and 7 yards respectively, how broad should the pillars be to have the same strength at a depth of 2000 feet, with the chambers and all other conditions remaining the same?

Ans.—In this question the depths are as 1 is to 4; therefore the pillars require to be 4 times stronger:

$$1 : 4 :: 7^3 : 1372; \sqrt[3]{1372} = 11 +,$$

requiring the pillars to be increased to 11 yards.

At a depth of 500 feet the pillar and chamber were $7 + 9 =$ 16 yards broad, but at 2000 feet the pillar and chamber will be $9 + 11 = 20$ yards broad; therefore if the pillar of 11 yards was to support 16 yards, it will have to be still further enlarged to support 20 yards, and the increase of weight is in proportion to the breadth.

$$16 : 20 :: 11^3 : 1663.75;$$

$$\sqrt[3]{1663.75} = 11.84$$

yards as the breadth of the pillar. If we now take this from 20 it shows what is left, for $20 - 11.84 = 8.16$ yards for breast,

which is nearly a yard too little. The exact quantity is found by the differential method of series. If we take one yard more width, or 21, and make the chambers 9 and pillars 12, there will be sufficient strength. After finding the breadth of pillar, throwing in another yard to cover the fractions is a simple and ready mode of solving the difficulty, and, although not mathematically correct, will be practically sufficient.

In this question the chambers have not been changed, but the pillars have. In the next question we will investigate as to how we can keep both chambers and pillars the same breadth and place the extra coal in reserve pillars at intervals, to break off the top and prevent squeezes while removing the pillars in the final mining.

Ques. 25.—In a seam 8 feet thick at a depth of 500 feet the chambers are 9 and the pillars 7 yards broad; how would you determine how much coal to leave in reserve pillars at a depth of 2000 feet, the chambers and common pillars to remain the same width or nearly so?

Ans.—In this proportion the first thing required is to determine the number of chambers and pillars to be placed between the large reserve pillars. In this case we will take the space of 5, which is

$$7 + 9 = 16, \text{ and } 5 \times 16 = 80$$

yards along the gangway in each panel. The depths are 500 and 2000 feet, or as 1 is to 4; therefore the weight on a given space is increased 4 times. The extra strength to suit the extra depth has to be all placed on the fifth pillar, and the other four pillars remain the same. If a pillar 7 yards broad supports 16 yards in width at 500 feet, it should support only one-fourth, or 4 yards, at 2000 feet to have the same weight on it; therefore the four pillars of 7 yards each will only support

$$4 \times 4 = 16$$

yards at 2000 feet deep. The whole width to be supported being 80, then

$$80 - 16 = 64.$$

Therefore there are 64 yards left for the fifth pillar of 7 yards to support; but the fifth pillar, like the others, can support only

4 yards at that depth. If it can support 4, and is required to support 64 yards, then as

$$\frac{64}{4} = 16,$$

the strength of the fifth pillar must be increased 16 times; therefore its breadth has to be increased, by proportion, as follows:—

$$1 : 16 :: 7^3 : 5488;\ \sqrt[3]{5488} = 17.6,$$

which is the breadth required for the fifth pillar. This is a total breadth for pillars of

$$4 \times 7 = 28, \text{ and } 28 + 17.6 = 45.6 \text{ yards.}$$

The whole breadth was 80, but the big pillar has taken up the space of one chamber; and as there are 5 pillars, there must be 5 chambers, which would require

$$5 \times 9 = 45$$

yards; this added to the breadth for pillars would be a total breadth of

$$45 + 45.6 = 90.6,$$

which places still more weight on the fifth pillar. We will therefore have to increase its strength still more to support the increased weight.

The four pillars each 7 yards broad will now support 16 out of 90.6 yards, leaving

$$90.6 \quad 16 = 74.6$$

yards to be supported by the fifth pillar. As it can support only 4 yards,

$$\frac{74.6}{4} = 18.65,$$

we must now increase the strength of the fifth pillar 18.65 times, instead of 16 times as before. This second proportion will be

$$1 : 18.65 :: 7^3 : 6396.95;\ \sqrt[3]{6396.95} = 18.5$$

as the width of the large fifth pillar; the whole width would therefore be—

For chambers $5 \times 9 =$	45
Four pillars of 7 yards =	28
Every fifth pillar,	$18\frac{1}{2}$
Total,	$91\frac{1}{2}$

is still too little, as it does not include the increase due to the restrengthened pillar, This is easily remedied by adding a yard. the method that mathematicians would use, is not intended for them. Although these applicable, they at least give us an idea of pillars, and as this is only a beginning this subject, it may lead to reflection in

perhaps the simplest method to determine the is to find by experience at what depth it is satis- say, 50 per cent. of the coal in the pillars to until the time comes for their removal, extra in the pillars for every 50 feet of Applying this rule to the last question following result:—

and the chamber 9 yards, there would be in the pillars at 500 feet below the surface. it would require to be increased as follows:—

$500 = 1500$; $1500 \div 50 = 30$; $30 + 44 = 74$

left at the increased depth. This on every 10 the gateway would be $100 - 74 = 26$ yards f chambers of 9 yards would be 27, or only o pillars of 7 would be 14 yards; $14 + 27 = 4$ 44 or 39 yards to be left in every third pill r, 118 in every sixth. This mode has the advant age and it leaves plenty for safety, some allowa ce being required to suit different thicknesses of the seam; but this plan distributes the coal, it by no means distrib- the strength.

What thi pillar wo consider suffi- to form a safe barr the workings of a on the Mountai el sill- full of water, and the ld dip 60°?

This is merely a from actual experience. The ans

lem are too varied, and the meaning of the word *safe* so indefinite, that it cannot be treated by any fixed law, or the thickness of the pillar measured out in feet. If we looked at this question merely in the aspect of a dam to keep out the water, we might approximate to a probable measurement for the pillar.

Assuming the lifts to be 300 feet, the total distance on the slant would be 900. The sine of 60° is 0.86602; $0.86602 \times 900 = 779$ feet equals the vertical height; $779 \times 62.5 = 48{,}718$ pounds per square foot as the water pressure at the bottom, which would diminish uniformly to zero at the top.

If we now imagine a horizontal column of coal one foot square of such length that its weight will equal the pressure at the bottom, and the pillar to taper to zero at the top, then such a pillar standing as a loose block would form a dam without any adhesion or support from the surrounding strata. A cubic foot of anthracite coal weighs 97 pounds; therefore, such a column would be about

$$\frac{48{,}718}{97} = 502 \text{ feet, or } 167\tfrac{1}{3} \text{ yards,}$$

which would be the width of base of a solid triangular block of the seam, which would form a dam from its mere weight. Practically, this pillar would be, say 60 yards wide at top. The average breadth of the triangular pillar is one-half the base, or 83 yards. If the top was made 60 yards broad, then the bottom would require to be reduced 60 to maintain the same quantity of coal in the pillar.

$$167\tfrac{1}{3} - 60 = 107\tfrac{1}{3} \text{ yards}$$

as the width at bottom of pillar. Therefore, according to this, the pillar should be 60 yards thick at top and widen out to 106 yards at bottom.

Another method of treating the question would be to consider the pillar as a wall to resist the total pressure by its mere weight, and allow the strength due to its solid position in the strata as a factor to counteract the effect of top weight and pressure from the action of the adjacent strata from under which the coal has been removed.

With a vertical height of 779 feet, the total horizontal pressure

on a vertical strip one foot thick through the width of the pillar would be equal to

$$779^2 = 606{,}841\ ;\ \frac{606{,}841}{2} = 303{,}420;\ 303{,}420 \times 62.5 = 18{,}963{,}750 \text{ lbs.}$$

total pressure on one foot thickness, or total pressure on a section of a seam one foot thick.

The resistance of a rectangular wall is its weight multiplied by the distance from the front surface to its centre of gravity, or one-half its thickness; and when the total pressure against it is known, its required thickness is found by the following formula:

$$T = \sqrt{\frac{2\ P}{3W}}$$

Where T = thickness of wall.
Where W = weight per cubic foot.
Where P = total pressure in pounds.
With coal at 97 pounds per foot this would be

$$\sqrt{\frac{2 \times 18{,}963{,}750}{3 \times 97}} = 362 \text{ feet} = 121 \text{ yards}$$

as the thickness of a rectangular pillar to resist the pressure by its mere weight. If this pillar was 50 yards at top, it would require to be

$$121 - 50 = 71, \text{ and } 121 + 71 = 192 \text{ yards}$$

at bottom to have the same amount of coal. We think if the coal was in average condition and the top broken and slid down as in ordinary cases, that a boundary pillar 50 yards thick at the top of the upper lift, 100 yards thick at the bottom of the third lift, would form a safe barrier to the colliery, as the word safe is understood in mines. The question of boundary pillars is one of great importance, and is likely to be of even more importance in the future. Where the conditions laid down in this question exist, the safety of the mine would be much increased if the main top was broken off from the boundary pillar.

To effect this, some pillars should be robbed out a short distance from the boundary to cause a general fall and break of top. When once a loose end is obtained the pillars next the

boundary can be taken out in succession, and the top being loose-ended will break off easier. When this is done any squeezing or weight will be disconnected and the boundary rendered practically safe as a barrier between the two collieries.

Ques. 27.—Suppose the abandoned workings of a mine are full of water, the seam being 12 feet thick of hard coal lying at an angle of 70°; if the vertical head of water is 500 feet, what thickness of chain pillar would be required below the water to insure the safe working of the chambers or breasts from below, the breasts and pillars being 9 and 7 yards wide respectively?

Ans.—A head of water 500 feet high exerts a pressure of

$$\frac{62.5 \times 500}{144} = 217$$

pounds per square inch or over 15½ tons per square foot. A pillar of hard coal across the top of a chamber 27 feet wide would not require to be over 5 yards thick to carry the weight; but as faults and soft slips occur in all coal seams, nobody would work or be allowed to work under such conditions, except for the purpose of tapping off the water.

CHAPTER VII.

MINE TIMBER—DAMS—MISCELLANEOUS.

Ques. 1.—Give your views on "propping" in general as practiced in mines, the method of distributing, placing in position, and paying for the work giving the best results for safety and economy.

Ans.—Statistics show that falling top is the greatest source of danger in coal mines. The transactions of mining institutes and kindred societies contain able papers on this subject. The conditions in mines, however, are so varied that a uniform system of propping is not practicable.

Props in lengths to suit the varying thickness of the coal seam are generally stacked convenient to the works. The standing of props being, as a rule, part of the miner's contract, he makes a selection from the props which are sent into his place and marks them with his ticket number. In some parts of the anthracite coal fields of Pennsylvania props 7 to 12 feet in length and from 8 to 12 inches in diameter are used. In such cases it is best to have the timber brought to the works in full length and each prop cut to fit the place prepared for it.

Another method is to have a given number of working-places under the care of a timber man, whose duty it is to prop and secure the top, the miner being only paid for mining and loading.

In England this deputy system has been largely adopted as the result of long experience in extensive mines, where the coal has to be transported long distances underground by complicated lines of haulage. Men specially trained and equipped for timbering can do it much cheaper and better for the same reason that division of labor in a factory [illegible] production.

The defect in this [illegible] as the [illegible] the top may suddenly beco[illegible]ici[illegible]

pend work and report to the "deputy." This induces him to take risks by waiting until his day's work is finished and he can report on his way home. Or the place may fall before the deputy can finish the work he has in hand, when he is apt to request the miner to stand a temporary prop to keep himself safe until he can get it properly secured. The miner will often comply rather than see his place fall, or lose time waiting; but knowing that it is something outside of his contract, he is apt to do the work, so that it may be as much of a trap as a security. We prefer the miner taking care of his own place for a given distance back from the face and holding him responsible.

Another feature of the prop question is that the mine officers cannot be expected to know every place where a prop is necessary, as the top may sound solid when examined and in a few hours become dangerous. The discharge of a shot may change all the conditions. To employ a force of men to sound or examine all the places liable to danger as mining proceeds would entail an expense not warranted by the present relation between the cost of mining and the prices obtained.

Another method is to stand the props at a given distance apart. Under some conditions this will give the best results as regards safety, but a large percentage of the timber is practically useless and the result in economy unsatisfactory. The rule of paying a given price per prop induces the miner to secure his place, but is apt to be abused. When work is slack he may stand props merely for the price paid. While the boss may see timber stood where he considers it unnecessary, he hesitates to order it stopped for fear of being blamed in case of accident. All systems of propping have defects. Rules that work well in one mine or seam do not give satisfaction in others, and good colliery management consists, among other things, in adopting the system best adapted to the nature of the top, the class of labor available, and such other conditions as may exist in the district where the mine is located.

Ques. 2.—Explain how to proceed in standing props, giving reasons, if any, why props are generally stood with the butt or thickest end next the top.

Ans.—This question naturally suggests itself from the fact

that columns in a structure are tapered from the base to the capital. The object of this is to give stability, and is simply copying nature's laws as shown in the taper of a tree.

To a certain extent props in mines ars pillars or columns supporting weight, but the conditions in a mine are so different from those in a building, that the law of columns cannot be applied to them.

Props in a mine are not intended to support the top; they are only expected to prevent loose pieces from falling unexpectedly and to act as indicators of danger.

When the top begins to press upon the timber the tightest props are forced into the cap-pieces, and begin to crack and splinter. The practiced eye and ear of the miner can thus detect the indications and notice these warnings in time to determine when it is no longer safe to remain in a properly propped working chamber. This kind of knowledge is the result of experience and observation.

In standing props a round foothold is cut deep enough to reach the solid ground, a handful of fine refuse is thrown in, and the prop rotated in the hole to bed it solid and give full bearing surface. The upper end is then driven along a piece of plank, or cap, placed against the top, being measured so that when driven tight it will be nearly at right angles with the dip.

The bottom of the prop is chipped off like the frustum of a cone, so as to make it fit easier and prevent splitting. This is most readily done at the smaller end, while the larger end gives more bearing surface on the cap-piece. Moreover, the butt end can generally be driven tightest, since being heavier it is less apt to spring or rebound with the hammer.

A prop is, therefore, not a column, but a brace, driven between the bottom and top, and the object in placing the big end up is to increase the surface against the cap, the bottom being firmly secured in the foothold.

It is important in timbering to have the ends of the props cut at right angles to their axis, the cap-piece bearing equally over the whole surface of the end and all the prop taking its share of the load.

The practice of driving a wedge over the end of the cap-piece

on the opposite side from the face, thus bending it down, may prevent props from being displaced by pieces flying from shots; but it tightens the prop on one side, and when weight comes on acts diagonally across the axis, rupturing the timber in detail, as is often seen in props splintered and crushed from one side.

The price of timber, the expense of the cutting and transportation, and the time consumed in putting it in position, make it of great importance to have it properly stood.

A limited number of props judiciously placed and properly stood do more good than mere numbers placed at random and carelessly fixed.

By proper attention to detail in this matter money can be saved to the operator and the safety of the operative increased at many of our mines.

If the weight acting on a prop were free, it would press vertically and the prop would have to stand vertical to resist it; but, practically, pieces that appear or sound loose are interlocked by the surrounding strata, so that their line of least resistance cannot be determined. The props are, therefore, stood nearly at a right angle to the dip, which brings the square-cut end to coincide with the cap and the thrust as far within the angle of repose as practicable.

Ques. 3.—What should be the difference in diameter between props for a seam of coal 6 feet thick and for one 12 feet thick, all other things being equal?

Ans.—If props were very short in proportion to their diameter, were straight in the fibre, had their ends flat and at right angles to their axis, were wedged so that every part of their cross-section took its due share of the load, then we would reason that they would be affected by the same law as columns in a structure, their strength varying nearly as the fourth power of their diameter, and inversely as the square of their length.

Props in a mine do not fulfil these conditions, and, therefore, the law for columns will not apply to them.

When the average prop takes the weight the effect is to bend it. This throws the thrust away from the line of the axis. The one side is ruptured by compression and the other by extension; therefore the causes at work destroying the prop are similar to

what occurs with a beam supported at both ends and loaded in the middle. We would, therefore, incline to estimate their relative strength by the same law.

This may not be theoretically correct, but in the present state of our knowledge we think it near enough for all practical purposes.

According to this law, the strengths of props would vary as the cubes of their diameters, and inversely as their lengths.

According to this proposition, the props in the 12-foot seam being double the length would, with an equal diameter, be only one-half the strength of those in the 6-foot seam; their *strength* would, therefore, have to be doubled.

If the relative strength changes as the cube of the diameter, then the diameters are proportional to the cube roots of the relative strength.

In this question the strength of the long props must be increased in the proportion of 2 to 1; therefore their diameters must be increased as the cube root of 2 is to the cube root of 1. The cube root of 1 is 1 and of 2 is 1.25+, requiring the long props to be one-fourth thicker. If the 6-foot props were 8 inches in diameter, then the 12-foot props would have to be—

$$\sqrt[3]{1} : \sqrt[3]{2} :: 8 : 10, \text{ or}$$
$$1 : 1.25 :: 8 : 10 \text{ inches, or}$$
$$1 : 2 :: 8^3 : 1024; \text{ and}$$
$$\sqrt[3]{1024} = 10+ \text{ inches}$$

as the required diameter.

Ques. 4.—If 7-inch props are the proper thickness in a seam 6 feet high, what thickness of prop would be sufficient where the seam thins off to 4 feet?

Ans.—In this proposition the diameter may be reduced, because

$$\frac{4 \times 7^3}{6} = 228.6, \text{ and}$$
$$\sqrt[3]{228.6} = 6.1+ \text{ inches},$$

as the required diameter.

Ques. 5.—Explain what is meant by double timber, its proper form, and application as generally required in coal mines.

Ans.—Fig. 54 represents a drift mine having the opening through the soft strata at the entrance secured by double timber. *ll* represents the legs, *c* the cap or collar, *m* the lagging or lining behind and over the timber, *R* the roof or top of the mine, *O* to *P* the coal seam, and *n* the notch or joint of the timber.

FIG. 54.

The different notches or joints adopted in double timber, round or square, are to some extent a matter of taste; but if the form conflicts with the laws relative to the strength and strain of materials, then we know that it must be wrong, regardless of all opinion.

It seems simple to make and stand double timber; but we have often examined long ranges of timbering with a large percentage of it imperfectly fitted. When a man pays for a piece of carpenter-work he insists upon it being properly fitted, and the same law should hold good in regard to mine timber. Any doubt on this point can be easily removed by examining the timbering in our average mines. The different forms of joint can be best explained by diagrams.

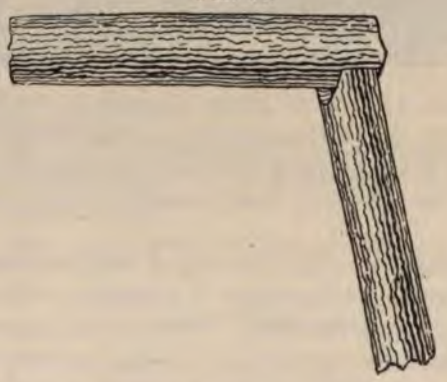
Fig. 55 represents a badly-fitting timber joint. The bearing being only at the front of the leg, the weight is thrown on so small a surface as to create a tendency to split the leg off. The shoulder of the notch being vertical, the part of the collar below

FIG. 55.

the face of the notch receives no support and, as the depth is the principal factor of strength, is apt to split off. This is a source of weakness which should be carefully avoided.

Fig. 56 shows the joint properly fitted. The face of the leg and the shoulder of the notch being slightly bevelled, give full bearing surface; pressure only binds it tighter.

FIG. 56.



We have heard objections raised, even by mine foremen, that with this form side pressure would have a tendency to force the leg inward from under the collar. Examination of the bevel

shows it to be only a few degrees from being square to side thrust, and so far within the angle of repose that, if the top is properly wedged, the leg will give way before being pushed inward.

Figs. 57 and 58 show another form of joint; but as it reduces

FIG. 57.

FIG. 58.

the bearing surface and cuts away the timber where it does the most good, we cannot see any reason for its adoption.

Figs. 59 and 60 show the form easiest made and giving the

FIG. 60.

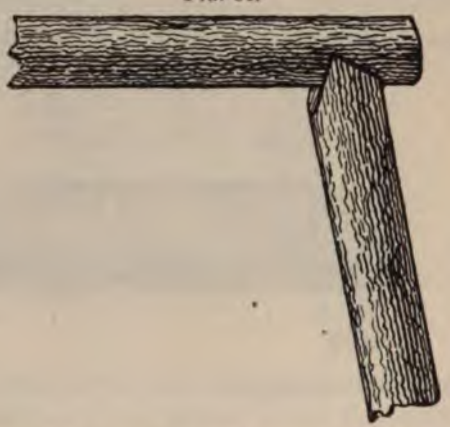
FIG. 59.

best results. It leaves the full strength of the timber and gives a full bearing surface to the end of the collar.

Fig. 61 shows another form. The notch is chopped out of the collar with an ax and the leg dressed off in the same way. This form is intended for securing the roof over mine tracks

where there are open spaces at the sides. The form of the notch prevents the legs from moving in the direction of the collar,

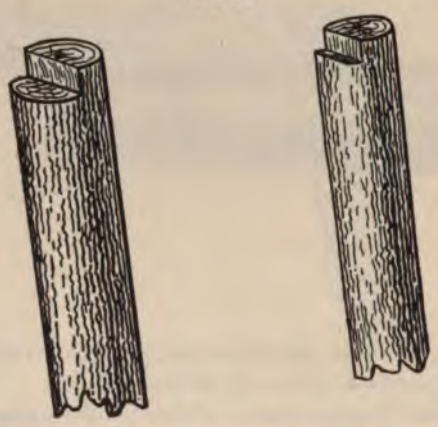
FIG. 61.



and stringers nailed across them from set to set hold them in the other direction.

Figs. 62 and 63 show another form of joint which is sometimes adopted, generally where the legs are stood nearly vertical.

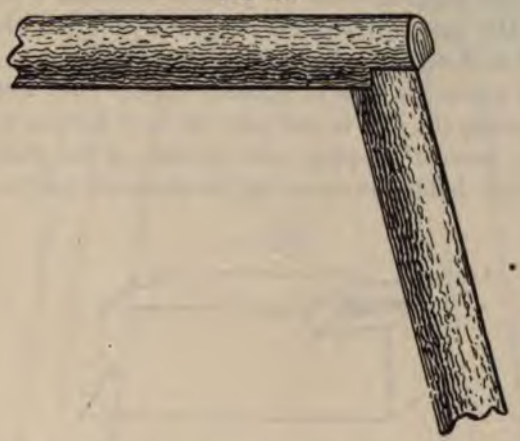
FIG. 62.



One objection to this form is the difficulty of fitting the double bearing so that both will bear equally. If the upper part of the

collar has to resist the pressure, it is apt to split off until it rests on the lower notch; and if the lower notch has to resist the

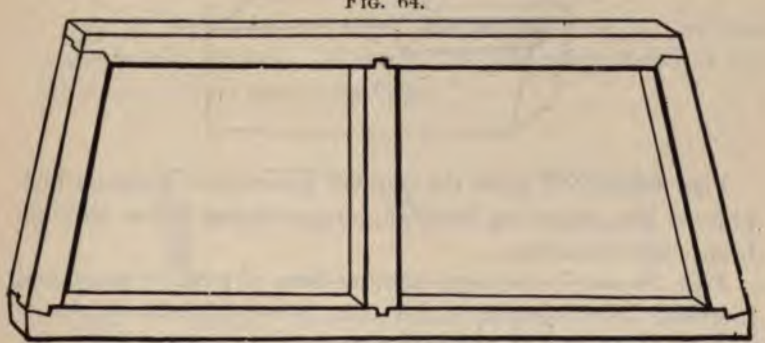
FIG. 63.



pressure, the front of the leg is apt to splinter. If properly fitted, this form is adapted for side pressure where the legs are not apt to push inward at the foot, and can be stood nearly vertical.

Timber in slopes for the entrance, and as far down as can be

FIG. 64.

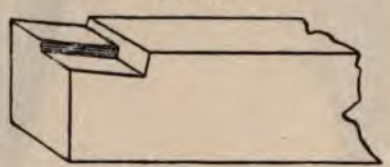


seen from the top, is generally squared, or at least slabbed, to present flat surfaces. This removes the useless weight of the

bark, and improves the general appearance of the work. It stands about 5° from being at right angles to the dip. The notching of the collar and foot sill are in two steps to prevent the legs from slipping down the pitch or the collar from slipping down off the legs. Where the bottom is solid the foot sill is dispensed with except a few setts at the entrance.

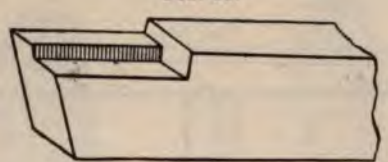
Fig. 64 represents a set of squared timber for a double track slope, showing the step in the joint at both bottom and top of the leg to prevent slipping, and the front of the wedge-shaped tongue which holds the centre leg for the same purpose.

FIG. 65.



Figs. 65, 66, 67, show the form of the notch in the foot sill, collar, and legs respectively. The check on the bottom of the legs at the front, is to prevent the bottom sill from slipping upwards from the pressure of the legs.

FIG. 66.



Figs. 68 and 69 show the dovetail groove and tongue which prevent the centre leg from slipping or being driven through from under the collar.

Figs. 70 and 71 represent another form of joint by tenon and mortice. The objection to this form is the difficulty in removing the legs when it becomes necessary to replace them with fresh timber, the tenon preventing the collar from being driven off the leg, or the leg from being driven out from under the

collar; generally, they have to be chopped out with the axe, rendering them unfit for any other purpose except blocks.

FIG. 67.

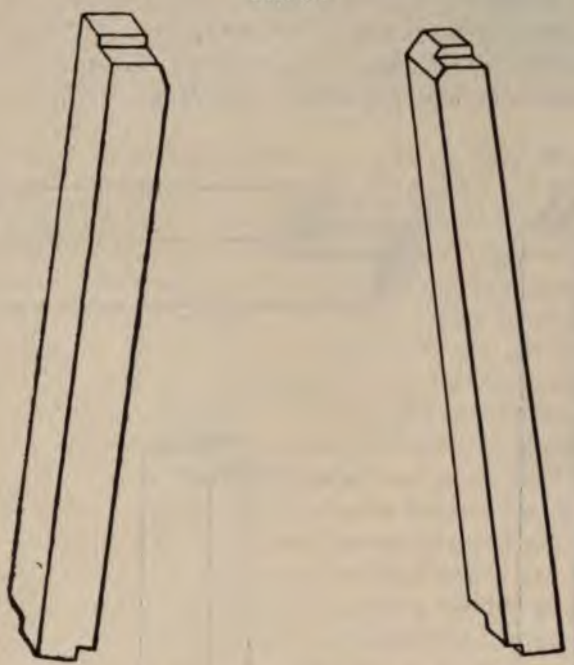
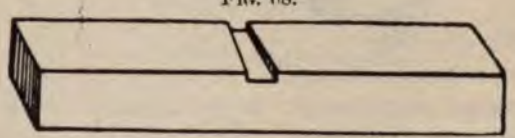


Fig. 72 represents the double notch for straight timbering with squared lumber. This form, as previously stated, is very satisfactory when accurately fitted.

FIG. 68.



Ques. 6.—Explain how to proceed, in placing the first timber in a slope, and advancing through the loose strata until solid roof is reached.

Ans.—The location being determined, an open cut is made to

the strata under the coal, and extending down until the full height of the timber is exposed as a face with the bottom sloping up to the surface, thus showing the dip and direction of the slope. Where the bottom under the coal is fire-clay, it is generally soft near the surface. Water and travel convert it into puddle, and in such places the first few foot-sills are apt to settle unevenly, and get slightly out of line, thereby marring

FIG. 69.

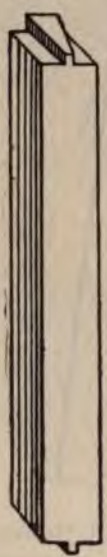


FIG. 70.

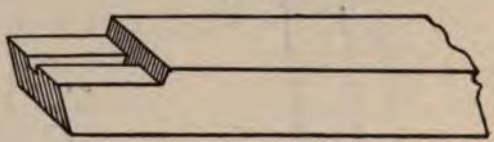


FIG. 71.

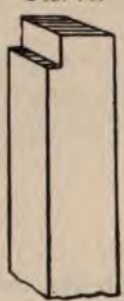
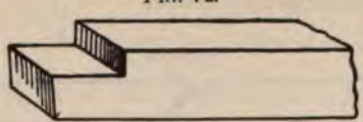


FIG. 72.



the appearance of the work. It is generally best in standing the first few sets of timber, to lay bed-plates under them; these are merely planks laid under each row of legs, across which the foot-sills are laid.

When the bed-plates have been levelled across, by means of a straight-edge, a foot-sill is laid at the face, the scribe-mark

through its centre is brought to coincide with the centre line of the slope and squared across it. The legs are set up, the collar lifted on, and the timber braced at 4° or 5° less than right angles to the dip. A plumb-line from the centre mark on the collar to the line on the bottom brings the legs to equal spread, and when properly made the joints will fit, and thus the first set is in position.

Other sets are placed as above and held in place by distance pieces, and tie straps nailed to the sill and legs. When 2 or 3 sets are stood they are lagged with plank and rammed with clay to exclude surface water. The frame of timber thus fixed gives shape to the future slope, and is a guide for height, spread, and direction. The next thing is to take out space for another set under the top. This is done by breaking in over the coal from the centre to each side, and temporarily securing the top by poles inserted up in the face, and running back over the timber. A space large enough for the next timber is thus taken out, another set stood, wedged, and lagged, and so on until the top becomes stronger and fore-poling can be dispensed with.

In sinking, the slope is kept in line by means of plumb-bobs hung from the centre mark on the collar or points are fixed in the roof from a short hole drilled up with a wooden plug driven into it. A round nail serves as a suspender for the plumb line.

Ques. 7.—If a collar in a gangway is 7 feet between the notches, and 10 inches in diameter of common hemlock timber, as used in mines, about how much weight will be required to break it under the following conditions?

1st. When wedged at the middle.

2d. When wedged along its whole length.

3d. When wedged at points one foot from each end.

Ans.—As before stated, the strength of timber to resist transverse strain, other things being equal, varies as the breadth into the square of the depth, which with a round or square section is as the cube of the diameter or side of the square.

The formula for determining the strength of square or round timber will, therefore, become

$$\frac{D^3 \times C}{L} = W; \frac{L \times W}{C} = D^3;$$

$$\frac{D^3 \times C}{W} = L; \frac{L \times W}{D^3} = C,$$

where L = length in feet, D = diameter or side of square in inches, W = weight in pounds, and C = a constant found by experiment.

The value of the coefficient C for square timber is the weight that has been found to break a piece of the specified wood one inch square, one foot between the supports, with the load applied at the centre. According to experiments by Tredgold, Barlow, Fincham, and others, this, for the weakest kind of fir, is about 440 pounds.

Mathematicians tell us that the value of C for a round stick, of diameter equal to the side of the square, is about ten seventeenths ($\frac{10}{17}$) of the value of C for a square stick. This fraction is partly due to reduction in area of the cross section, and partly due to the fact that the material in the round stick is more concentrated about the plane of the *neutral axis*. The value of the constant for square timber must be reduced in that proportion, as follows:—

17 : 10 :: 440 : 259 pounds nearly.

These results being obtained with perfect specimens, under the usual favorable conditions attending experiments, we do not expect such results in practice.

Timber when wet, as is often the case in mines, loses half its strength.

The timber usually used in mines is not seasoned to develop its strength, is often cut too green, or over-ripe, and much of it is twisted and knotty.

This may reduce its strength nearly one-half, so that for mine timber the coefficient may be reduced two-fifths, or say 150 pounds, which in the absence of actual tests, we think near enough for our present purpose. Applying these figures to our question, we obtain the following result:—

$$\frac{10^3 \times 150}{7} = 21{,}428$$

pounds, or 10 tons 14 hundred-weight and 28 pounds, as the breaking weight, with the load acting at the middle.

If wedged along the whole length and the load distributed from end to end, the strength will be doubled, and the above result may be multiplied by 2, or

$$2 \times 21{,}428 = 42{,}856$$

pounds, or 21 tons 8 hundred-weight and 56 pounds.

With the wedging one foot from each notch, the weight on the collar would be thrown two-sevenths of the whole length from the supports; therefore the weight required to break the collar under such conditions would be equal to the breaking load of a beam only two-sevenths as long as the whole beam. The reason for this is that the breaking moment of the beam is one foot multiplied by half of the load, or the load at one wedge, which is just the same moment as is given by a beam 2 ft. long and loaded in the middle. Therefore,

$$\frac{10^3 \times 150}{2} = 75{,}000 \text{ pounds.}$$

or $37\frac{1}{2}$ tons, as the breaking load under these conditions.

The three results thus obtained are—

First	Wedged in the middle	10.71 tons.
Second.	Wedged along the whole length . . .	21.42 tons.
Third.	Wedged at points one foot from each end	37.50 tons.

These figures are on the assumption that the pressure is vertical, and are of practical value as showing relative resistances of collars when the weight is thrown on different parts, by lagging, cribbing or wedging. Much material has been wasted and labor lost by neglecting the simple principle illustrated by this question.

Ques. 8.—If you were timbering a gangway with collars 7 feet long and 10 inches in diameter, and required to widen it for a side track, the collars on which would be 14 feet long, how would you increase the length and diameter of the collars over the switch and the diameter over the side track?

Ans.—Assuming the length from the switch-points under the 7-foot collar to the frog to be 16 feet, and to where the tracks were 4 feet apart to be 9 feet more, we have a length of 25 feet

from the 7- to the 14-foot width. With timbers 5 feet between centres, there would be 4 sets and 5 spaces between the last 7-foot and the first 14-foot collar, the difference to be gained on the length of the collars being 7 feet,

$$\frac{7}{5} = 1.4 \text{ feet},$$

therefore each of the four collars must be increased as follows:

$$7 + 1.4 = 8.4, \quad 8.4 + 1.4 = 9.8, \quad 9.8 + 1.4 = 11.2,$$
$$11.2 + 1.4 = 12.6$$

feet, as the length of the four intermediate collars respectively.

The weight on the collar increases and its strength decreases directly as its length. Therefore, to find the relative diameters we have the compound proportion,

First collar,

$$\left.\begin{matrix} 7 \text{ length} \\ 7 \text{ weight} \end{matrix}\right\} : \left.\begin{matrix} 8.4 \text{ length} \\ 8.4 \text{ weight} \end{matrix}\right\} :: 10^3 \text{ diameter} : \text{diameter}^3,$$

$$= 7^2 : 8.4^2 :: 10^3 : 1440,$$

and $\sqrt[3]{1440} = 11.2$ inches, as the diameter of the 8.4-foot collar.

$$7^2 : 9.8^2 :: 10^3 : 1960,$$

and $\sqrt[3]{1960} = 12.5$ inches, as the diameter of the 9.8-foot collar.

$$7^2 : 12.2^2 :: 10^3 : 2560,$$

and $\sqrt[3]{2560} = 13.6$ inches, as the diameter of the 11.2-foot collar.

$$7^2 : 12.6^2 :: 10^3 : 3240,$$

and $\sqrt[3]{3240} = 14.7$ inches, as the diameter of the 12.6-foot collar.

Applying the same rule to the 14-foot collar gives a diameter of

$$7^2 : 14^2 :: 10^3 : 4000,$$

and $\sqrt[3]{4000} = 15.8$ inches over the double track. The 14-foot collar being double the length of the 7-foot, if the weight remained constant, the diameter would have to be

$$\frac{2 \times 10^3}{1} = 2000.$$

and $\sqrt[3]{2000} = 12.5 +$ inches; but with the width of the place doubled, if the top is affected by the same law, we would expect the roof to weaken, and the weight upon the timber to increase as the width, the same as the resistance of the timber decreases.

Therefore, on general principles we would reason that because of the double span a 14-foot collar would have to support double the weight of one of 7 feet.

On this assumption the 14-foot collar would have to be 4 times as strong; twice for the double length, and twice for the double weight, or,

$$1 : 4 :: 10^3 : 4000,$$

and $\sqrt[3]{4000} = 15.8$ inches diameter, the same result as found above, or 16-inch timber.

Ques. 9.—If you were driving a gangway $7\frac{1}{2}$ feet high, 6 feet wide on top, and 10 feet wide on bottom, double-timbered, the collars being of 10-inch and the legs of 9-inch timber respectively, and the airway parallel to the gangway 6 feet high, $4\frac{1}{2}$ feet wide on top, and $7\frac{1}{2}$ feet wide on the bottom, what sized timber would be sufficient for collar and legs in the airway, the condition of the coal of the top and the distance from centre to centre of timber being the same as in the gangway?

Ans.—If the gangway collar is 6 feet long and 10 inches in diameter, an airway collar $4\frac{1}{2}$ feet long would, all other things being equal, require a diameter of

$$6^2 : 4.5^2 :: 10^3 : 562\ 511,$$

and $\sqrt[3]{562.511} = 8.2$ inches. A gangway 6 feet wide at top, and 10 feet at the bottom has a spread of 2 feet. If the height is $7\frac{1}{2}$ feet to the rail, then the length of the leg to the same level is

$$\sqrt{(7.5^2 + 2^2)} = 7.7 \text{ feet},$$

and allowing length to sink the leg to the bottom of the drain ditch would require it to be, say, 9 feet. With the airway $4\frac{1}{2}$ feet wide at top, and $7\frac{1}{2}$ feet wide at bottom, we have a spread of $1\frac{1}{2}$ feet; and as the height is 6 feet, the length of the leg will be

$$\sqrt{(6^2 + 1.5^2)} = 6.18 \text{ feet}.$$

Now allowing 10 inches for the foothold, the length would be 7 feet, which gives us the following proposition:—

If a leg 9 feet long and 9 inches in diameter is correct for the gangway, what should be the diameter of a leg 7 feet long in the airway to resist the same conditions?

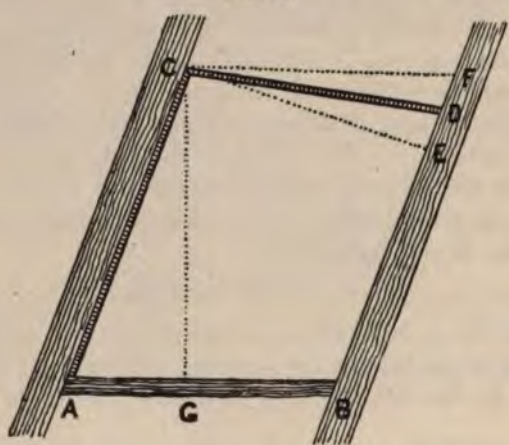
$$9 : 7 :: 9^3 : 566,$$

and $\sqrt[3]{566} = 8.27$ inches as the correct diameter for the legs in the airway.*

Ques. 10.—If you were timbering a gangway with "post and bar," in a seam of coal 8 feet thick, dipping at an angle of 70°, the post being 8 feet long, and assuming a load of 8 tons to each set, state what inclination you would give to each timber, and show what amount of strain would be thrown on each.

Ans.—For the purpose of investigating this proposition, let *A* and *B*, in Fig. 73, be the top and bottom of the seam re-

FIG. 73.



spectively. *AC* the post 8 feet long and *CD* the bar. Now as all the space available is required for a gangway in a seam 8 feet thick dipping 70°, we would give the post the same inclination as the top. Placing the bar at right angles to the dip would make it shortest and strongest for the same sized timber as the line *CE*, but it would require a step-hole cut in the bottom slate at *E* to secure it. This step-hole, however, might break off and throw the timber loose. Placing it as the line *CF* would rest it against the slant bottom, only requiring the end bevelled to fit the rock, and stamping to prevent shots from

* This makes no allowance for the decreased width of the airway. F. E. B

displacing it sideways. This plan would make the timber longer and consequently weaker. We would, therefore, give the bar an inclination between level, and being at right angles to the dip as represented by the line *CD*, thus reducing the length of bar as much as is consistent with giving it a solid foothold against the bottom. Because the angle of dip *BAC* is equal to the angle *CFB*, the angle *CFB* is 70°, and because the angle *CEF* is a right angle, it is 90°, therefore the difference between them, *ECF*, is 20°; if we bisect 20° by the line *CD*, its inclination will be 10°. Therefore, we would give the bar *CD* an inclination of about 10 degrees.

By the terms of the proposition the bar has a load of 8 tons. With one-half supported on the bottom rock, 4 tons would be supported by the post; but because the post stands at an angle of 70°, the thrust is as much greater than the load as *CA* is greater than *CG*.

Because *CA* is the hypothenuse = 8 feet, and the angle *GAC* = 70°, if we multiply 8 by the sine of 70° we find the length of *GC*—

$$8 \times 0.93869 = 7.5 \text{ feet; then}$$
$$7.5 : 8 :: 4 : 4.26,$$

or about 4¼ tons, as the strain on the post.

With the above problem applied to a structure where the magnitude and direction of the strains could be determined, a correct solution might be reached; but the vagueness of its terms renders such a result impracticable.

This question was one of an examination for inspector of mines in the anthracite coal region of Pennsylvania, and is inserted here to show what questions boards will sometimes ask.

Ques. 11.—Explain the object and effect of standing the legs of timber with a *spread*, or wider at the bottom.

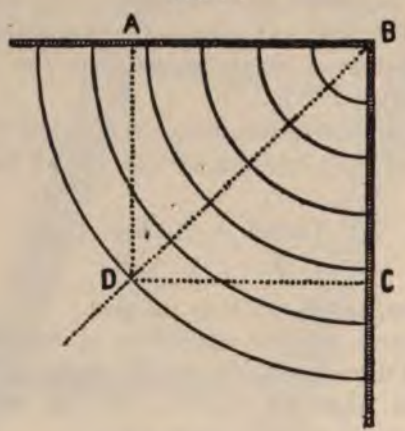
Ans.—The reason for spreading double timber at the bottom is that a gangway wider at bottom than at top gives more space for the car-track, the drain-ditch, and the handling of cars without increasing the span of the roof so as to weaken it. It also prevents side pressure from forcing the legs inward at the bottom; therefore timbering with a spread gives the most space

consistent with a short span on top. Timber is often stood with too much spread, from a mistaken idea of being better adapted to resist side pressure.

If the pressure is vertical, slanting the legs increases the thrust. If the pressure comes from both top and side, then the resultant of both acts in a line bisecting the angle formed by the collar and leg, and increases as the angle increases.

Let Figs. 74 and 75 represent two sets of timber, one stood square and the other slant. Assuming the pressure from both top and side, so that its resultant will be 6 tons in the direction of the arrows, at *B*, then the six equal divisions on the line *BD*

FIG. 74.

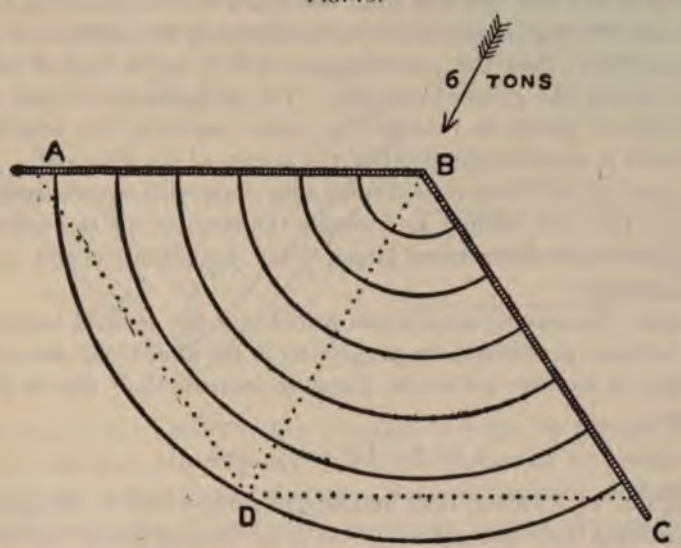


will represent in magnitude and direction the resultant of these forces. By the parallelogram of forces, the equal divisions on the lines *AB* and *BC* represent the tons of strain on both collar and leg. On the square timber it is about 4¼ tons on each, or 8½ tons; while on the slant timber it is about 6¼ tons on each, or a total of nearly 12½ tons. This shows the importance of standing timber as nearly square as is consistent with preventing the legs from being pushed inwards at the bottom. Timber has a better appearance with some spread, it having the same effect upon the eye as taper on a column.

With a clear height of 7½ to 8 feet we would give the legs a

spread of about $1\frac{1}{2}$ feet, or in that proportion, except for special conditions.

FIG. 75.



Ques. 12.—Suppose you had round timber, 20 inches or so in diameter, which you were slabbing on a saw-mill for collars in a slope: what shape would you saw them to get the strongest squared collars out of the round log?

FIG. 76.

Ans.—We would cut them according to Fig. 76.

Let the circle *ABCD* (Fig. 76) represent the end of the log.

Draw the diameter AC and divide it into three equal parts, in SS. From these points on opposite sides of AC raise perpendiculars, SB and SD, and from their points of contact with the circumference at B and D draw the lines AB, BC, CD and DA, respectively; then the parallelogram $ABCD$ is the form of section giving the greatest strength. The proportion of depth to breadth is about as 7 is to 5, or, more correctly, the breadth squared is equal to one-third of the square of the diagonal.

Ques. 13.—If you were timbering a slope with square timber $12'' \times 12''$, and wished to increase the strength of the collars one-half by making them larger, what dimensions would you make them?

Ans.—Increasing their depth would give the greatest amount of increase in strength in proportion to the additional size and weight of timber; we would therefore increase their size as follows:—

$$1 : 1.5 :: 12^2 : 216, \text{ and}$$

$\sqrt{216} = 14.7$ inches, thus making the collars 14.7×12 inches to increase their strength as 3 is to 2, or one-half more, in order to resist vertical pressure.

Ques. 14.—Suppose you were timbering a slope with collars 15 inches square; by taking 3 inches from the width and putting it on the depth, still keeping the collar the same weight, how much stronger would it be to resist top weight?

Ans. —Taking 3 inches from the breadth would weaken it as 4 to 5. The $15 \times 3 = 45$ square inches would be $45 \div 12 = 3.75$, which would increase the depth to $15 + 3\frac{3}{4} = 18\frac{3}{4}$ inches. This would increase the strength as 18.75^2 is to 15^2, or as 1 is to 1.55. It was decreased as 1 is to 1.25, and $1.55 \div 1.25 = 1.24$. Therefore the actual increase in strength has been 24 per cent., or nearly as 4 is to 5.

Ques. 15.—Make diagrams showing some of the forms of joint used in shaft timbering.

Ans.—Shafts being rectangular the timber is merely frames placed horizontal and held at intervals by distance-pieces spiked in the corners.

These frames are lagged or lined behind by short pieces of

strong plank running vertically between each frame and filled in tight to secure loose strata from falling.

Figs. 77, 78, 79, 80 show the mitre and half-check joint used for the corners of short frames. Sometimes a gain is cut in

FIG. 77.

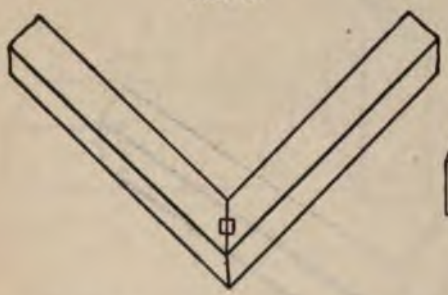
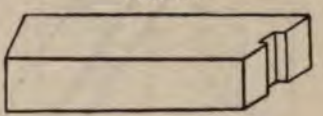


FIG. 78.



the face of the bevel for the insertion of a key, to prevent the joint-faces from slipping out of line till tightened by pressure. But if not properly fitted this has a wedging tendency, and may result in splitting; therefore we are not in favor of the practice.

FIG. 79.

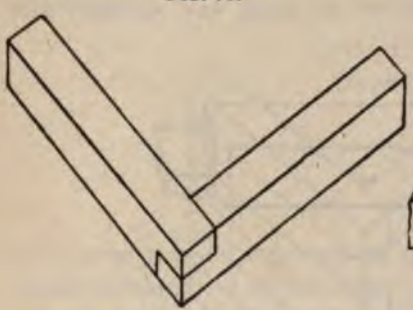
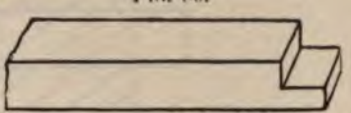


FIG. 80.

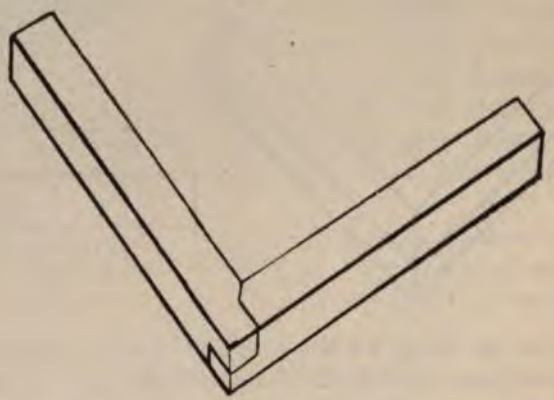


Figs. 81 and 82 show a form of joint combining the principles of both, being at once a mitre and a check, and if properly made will fill the requirements better than either. No matter what the form may be, wedging tendency to split should be avoided, as shaft timber may be subjected to great pressure.

In large shafts, divided into 4, 5, and even 6 compartments, the side timbers cannot be got or handled full length, and have

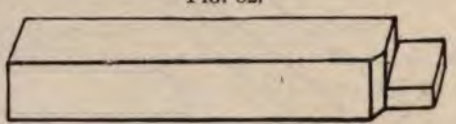
to be in sections, joined half on the end of the buntons or cross-pieces, which separate the compartments, the joints *breaking band* on different buntons.

FIG. 81.



In some cases the buntons are tongued and grooved to prevent them falling through, as shown in Figs. 83, 84, 85, 86, and sometimes only squared, driven in tight, and fastened with spikes.

FIG. 82.



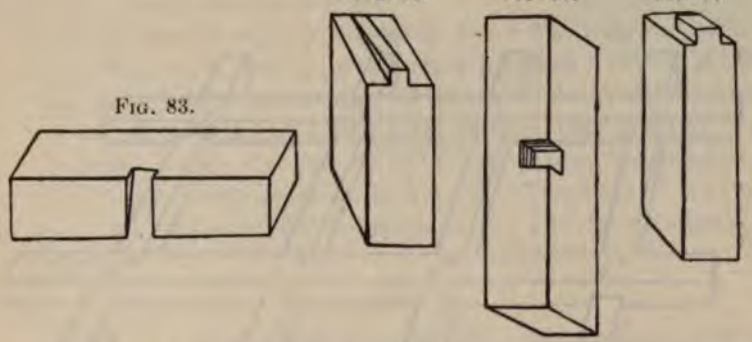
In sinking through mud or loose ground likely to cave in and produce great pressure on the shaft, the timber is put in skin to skin, or what is know as *cribbing*, with the ends of the logs butted alternately, as shown in Fig. 87, the corners being bound by a triangular stringer spiked to the logs, as shown in the figure.

For the purpose of keeping out water or fine sand or mud, cribs are sometimes built of plank, laid on in layers, with tar-cloth joints, and pinned down, plank by plank, on the same principle as wooden ships are built. Some shafts have double

cribbing and the space between the walls rammed with stiff clay puddle to exclude water.

Fig. 84. Fig. 85. Fig. 86.

Fig. 83.



Ques. 16.—Describe how you would proceed in placing the first few sets of timber in commencing a shaft.

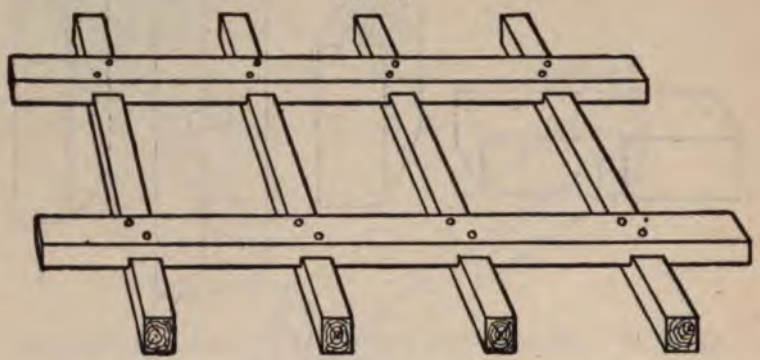
Ans.—To describe in detail all the different plans that have been tried to overcome difficulties encountered from mud beds, loose gravel, and water would be too voluminous for our present purpose.

Fig. 87.

The location having been determined, the loose surface is removed down to the solid. Upon this floor is placed the bed-frame or first timber.

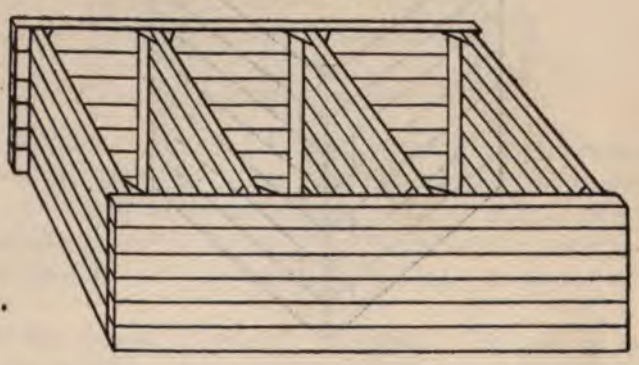
This is a frame of logs jointed together much in the form shown in Fig. 88. This frame being levelled and properly bedded forms the *shaft proper*. The logs projecting on to the

FIG. 88.



ground round the shaft are for the purpose of suspending the first few sets of timber until they are wedged and packed solid. When the ground has been dug out inside this frame as deep as it will stand, the next timber is put in, and so on. A very convenient plan is to have auger holes bored through the tim-

FIG. 89.



bers at each end near the corners, and long bolts run loosely through them for hanging the timber until it is wedged and lagged.

These hanger bolts have a nut on each end. When the timber is hung it can be easily levelled by adjusting it with the nuts; and the bolts being $\frac{1}{2}$ inch less diameter than the hole are easily removed for the next set.

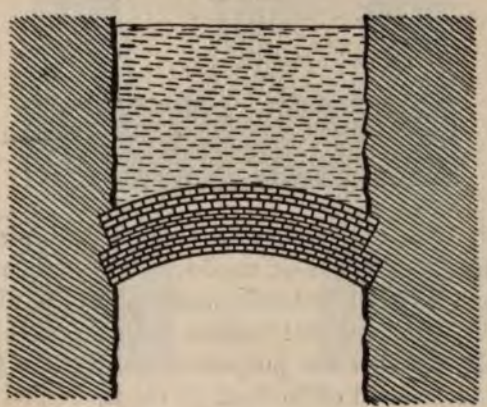
Shafts from 12 to 18 feet long by 6 to 8 feet wide are sometimes lined with *barring* or planking 3 to 5 or 6 inches thick. The sides are long planks the full length of the shaft, the ends and partitions being fitted in tightly between them, and having triangular corner stringers firmly spiked inside the corners of each compartment, as shown in Fig. 89. This form of timbering is well adapted for ordinary shafts, and is considerably cheaper.

DAMS.

Ques. 17.—How would you construct a dam in a gangway or tunnel to support a head of water 1000 feet high?

Ans.—By building arches of stone laid in cement, with the

FIG. 90.

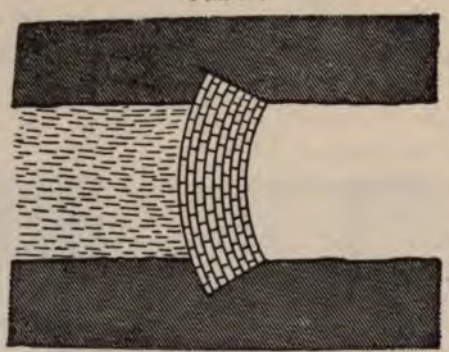


convex side facing the pressure. The site for such dams should be carefully prepared, and the skew from which the arch springs should be cut with the pick, because plugging and blasting are apt to open cracks in the strata and cause leaks. In a common tunnel or gangway such dams are generally built by arching from side to side, as in Fig. 90; but if the place where the dam

is to be built is very wide in proportion to its height, then the arching is from the bottom to top, as shown in Fig. 91.

Dams to resist pressure can also be constructed of timber. The site for a timber dam is prepared in the same manner as for stone or brick, only the sides have to be dressed off much smoother in order to present a straight surface on which to bed the timber. The arch is built of squared billets, cut tapered to coincide with the radius of the arch. The first layer must be bedded with cement; then each successive layer is spiked together, with tar-cloth joints between them. A man-hole is left in the bottom for the workmen. When the dam is built the seams on the inside are driven full of pine wedges; every

FIG. 91.

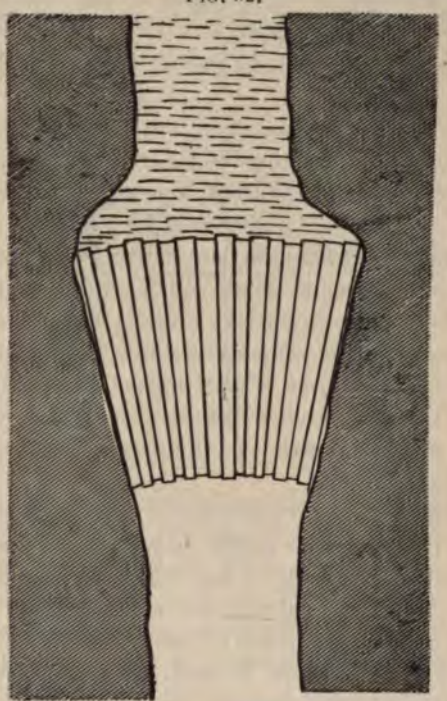


crevice round the wall is also driven tight with pine splinters until the dam forms almost a solid wedge. When completed the man-hole is closed by blocks made to fit tight like a wedge, the key block being a little longer, with a long eye-bolt run through its length, for the purpose of pulling it in tight. The skew sides are dressed off in front of the timber, so that if the whole dam is forced forward by the pressure there will be no notch to prevent it wedging itself tight. Such dams will resist great pressure (see Fig. 92), but are not so conveniently erected as brick, because brick-work is much more easily adapted to the uneven surfaces generally encountered in such work. If the timber is put in dry and well wedged, the water and pressure will swell and drive it tight; but timber dams of this kind are

not suited for large openings, and in some situations it is impracticable to obtain a surface to which they would fit watertight. We would therefore prefer to build dams of brick or stone for resisting heavy pressure, like those represented in Figs. 90 and 91.

When a dam is to be erected to resist a head of water 1000 feet high, the first thing is to inspect the locality and locate the

FIG. 92.



dam in the solidest place available. The cutting out of a site and plan of the dam is very simple. If we know the vertical height to which the water will rise, we can calculate the pressure the dam will have to resist, because water on a dam is a steady load, uniformly distributed over the whole exposed surface. The next thing we require to know is how thick should we make the arch to resist the known pressure and allow sufficient mar-

gin for safety, and of what material it should be built. We should also know the strength of the material of which the dam is to be constructed, and as these dams will practically be brick, it is with that material we are principally interested. According to the tables given by Eytelwein, the weight necessary to crush pillars of brick is 1000 pounds per square inch, or 144,000 pounds, equal to 72 tons per square foot; and that common fir timber, such as would be used to build an arched dam, when subjected to crushing at right angles to the fibre will require a force of 500 pounds per square inch, or 72,000 pounds, equal to 36 tons per square foot, to indent it to a depth of $\frac{1}{10}$ of an inch.

We now come to the dam, and the question arises, when we wish to calculate the thickness of an arch necessary to resist the pressure, whether we are to treat it as a *girder* or a *pillar* under compression.

It is evident that an arched dam which forms a short segment of a large circle resists pressure much on the same principle as the upper flange of a girder, the material being all on the compression side of the *neutral axis* and subjected to thrust; but our knowledge of the subject is too limited to determine whether this or the "theory of arches" is the proper guide in calculating the strength of such dams. In the present state of our knowledge we prefer the latter.

By the terms of the proposition the dam is to be erected in a gangway or tunnel. The width may therefore be taken at 12 feet. Assuming the arch to be from side to side, then the height, within the limits of practice, will not affect the question.

A dam arched across a place 12 feet wide can be either a large segment of a small circle or a small segment of a large circle; and, as small circles are stronger, we at once see that the length of the radius of the arch is important.

According to the theory of arches, when the load on an arch is a uniform normal force, as on a dam, then the line of thrust is along the circle, and its resultant is found by the parallelogram of forces.

Let Fig. 93 represent the dam 12 feet wide with a radius of 8.48 feet or $8\frac{1}{2}$ feet nearly. $(12 \div \sqrt{2} = 8.48)$

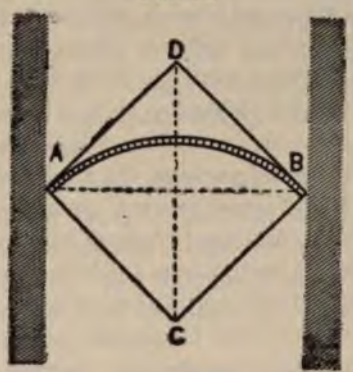
The pressure of 1000 feet of water column is $62.5 \times 1000 = 62,500$ pounds, or $62,500 \div 2000 = 31.5$ tons per square foot. Now the thrust on this dam is equal to the pressure multiplied by the radius. The dam is 12 feet wide, and a square whose diagonal is 12 feet has a side equal to

$$12^2 = 144, \frac{144}{2} = 72,$$

$$\sqrt{72} = 8.48+ \text{ or } 8.5 \text{ feet.}$$

Then the pressure per foot into the radius is 31.5 tons $\times$ 8.5 =

FIG. 93.



267.75 tons, the total thrust along the central line of the arch. This will be a thrust of 267.75 tons at a and b each. If we now, by the parallelogram of forces, call each of the tangents ad and bd equal to a force of 267.75 tons, then the diagonal dc will balance both. Again, because the parallelogram $adbc$ is a square, the angle at d is a right angle. Therefore $(ad^2) + (bd^2) = dc^2$, which gives the value of the diagonal dc as follows:

$$267.75^2 \times 2 = 142,200,$$

$$\sqrt{142,200} = 378 = 12 \times 31.5+ \text{ tons}$$

as the total thrust on each foot of height of dam which proves we are right.

The ultimate resistance of brick to crushing is given at 72 tons per square foot. Now if we tax the resistance of the brick to

say $\frac{1}{5}$ as a factor of safety, the available strength of the brick would be

$$\frac{72}{5} = 14.4.$$

This being the amount of active pressure to be absorbed by each foot of thickness in the dam, therefore it has to be

$$\frac{267.75 \text{ tons}}{14.4 \text{ tons}} = 18.5 \text{ feet.}$$

as the thickness of arch required for such dam. This is on the assumption that the whole thickness of the brick-work will come into play and bear its full share of the work. In practice this will not hold in a structure of such thickness as 18.5 feet. The brick on the outside would add very little to the strength. The brick next to the pressure would have to resist the force, and if it was partly elastic and yielded so as to let every brick take its due share of the pressure, in successive layers back through the dam, then the dam would perform the work; but brick will not do this. If brick will crush with a pressure of 72 tons per square foot of section, and the pressure applied is 267 tons, or more than 3 times, no matter how thick the dam might be, the pressure might crush the brick of the inside layers, and destroy the dam in detail from the inside out, and the only cure for such conditions will be to use stronger material.

If we took good stone which will require a pressure of 5000 pounds per square inch, or 720,000 pounds per square foot, to crush it, we would have a safer dam. 720,000 pounds ÷ 2000 = 360 tons, and taking $\frac{1}{5}$ as a factor of safety, we would have an available strength with good stone of 360 ÷ 5 = 72 tons per square foot. Dividing the thrust along the arch by this strength would give

$$\frac{267}{72} = 3.7 \text{ feet}$$

as the required thickness of a stone arch. To establish confidence and prevent leakage, the thickness should be more than doubled. We would, therefore, build the dam of good hard stone cut to the proper shape for the arch and jointed with cement, the stones as hard as practicable, and the arch 10 feet thick.

For pressures below 600 feet, brick might be sufficient. Where brick dams have to resist very high pressures, it has been suggested, and even put in practice, to erect a relief dam outside of the main one and load it to perhaps $\frac{1}{2}$ the height where the conditions admit of it being done; but if the outside material of the dam cannot be brought into play to take its share of the work by having it transmitted through the material, we would naturally suppose that the relief dam would not prevent the excessive pressure from rupturing or crushing the brick-work in detail from the inside outwards.

This source of weakness in structures has been practically demonstrated by the bursting of large guns, where the metal was of such thickness that it could not absorb its share of the strain. The force of the explosion fatigued and destroyed the metal from the core outwards, causing the gun to burst, after being thoroughly tested.

Ques. 18.—Suppose that after heavy rains, or when a thaw in spring increases the quantity of surface water until it threatens to overpower the pumps, and to relieve them you build a dam in one of the gangways as a temporary sump or lodgment for the water: If the dam is 10 feet wide and 6 feet high, the water having reached the top of dam:

First. What will be the pressure per square foot on the bottom of the dam?

The pressure per square foot at the bottom is equal to

$$6 \times 62.5 = 375 \text{ pounds}$$

per square foot. This is the pressure at the extreme bottom, but at the top of the first square foot it is only

$$5 \times 62.5 = 312.5 \text{ pounds per foot;}$$

the average of $375 + 312.5 = 687.5$,

$$\frac{687.5}{2} = 343.75 \text{ pounds.}$$

as the pressure per foot on a bottom plank 12 inches broad.

Second. What will be the total pressure in pounds against the dam?

The pressure per foot at the bottom being 375 pounds and at

the top zero, the average pressure per foot against the whole dam is

$$\frac{375}{2} = 187.5 \text{ pounds}$$

per square foot. The dam is $6 \times 10 = 60$ square feet; therefore, the total pressure on the dam is

$$187.5 \times 60 = 11{,}250 \text{ pounds.}$$

The same result may be found by taking one-half the square of the height into the breadth into 62.5, or,

$$\frac{6^2}{2} = 18;\ 18 \times 10 \times 62.5 = 11{,}250 \text{ pounds.}$$

Third. Where would you place a horizontal bar across the dam to support it so that the pressure above the bar would balance the pressure below it?

In order that the total pressure above and below the bar should balance each other, the bar would require to be at the centre of pressure, and as the centre of pressure on a vertical dam is $\frac{2}{3}$ the depth from the surface of the water, I would place the bar 2 feet from the bottom of the dam.

Fourth. What thickness would a common hemlock plank 12 inches broad require to be, if placed across to form the bottom of the dam?

The thickness of plank necessary to resist the pressure is found by the formula for determining the strength of beams, or

$$\frac{\text{Length in feet} \times \text{total load in pounds}}{\text{Breadth in inches} \times \text{constant}} = \text{thickness in inches}^2.$$

The value of the constant being the number of pounds that will break a piece of fir one inch square, one foot between the supports, with the load applied in the middle.

In this proposition the load is distributed the whole length. As beams will support double the weight if distributed the whole length; therefore, we must calculate the thickness for only one-half the load as applied at the middle. The pressure along the bottom plank was shown to be 343.75 pounds per square foot, and the dam is 10 feet wide; therefore, the total pressure against the plank is $343.75 \times 10 = 3437.5$ lbs.

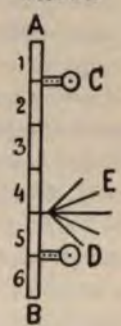
$$\frac{3437.5}{2} = 1718.75 \text{ pounds} = \tfrac{1}{2} \text{ load.}$$

The constant for fir is about 350 pounds, then by the formula the square of the thickness will be

$$\frac{10 \times 1718.75}{12 \times 350} = 4.0922, \text{ and } \sqrt{4.0922} = 2.022 \text{ inches}$$

as the required thickness of plank; but as this is the ultimate strength, we should allow a margin of safety by making it 4 times as strong as the calculated resistance; as the strength varies as the square of the thickness. If we double the thickness, it will make it 4 times stronger; therefore we would make the planking 4 inches thick. Practically, the dam will receive a great deal of support from being sunk into the bottom slate to make it water-tight. The planking above the first plank have less and less weight against them to the top. and if the structure is bound together properly to make it water-tight, the upper planks will take some of the load at the bottom, and make the whole resistance of the dam still greater, so that a dam of double 2-inch plank will be amply strong without any props or braces except to secure the ends. The great practical difficulty with such dams is to prevent leakage. To make them water-tight, generally requires a trench cut in the bottom slate and continued up the sides. The dam is often built by standing 3 or 4 props in line with the trench, across the gangway, then nailing 2-inch planks on both sides and puddling the space between the planks with stiff clay. Sometimes a permanent dam is placed across a gangway with a movable door over the car track, which can be quickly placed to dam water in case of accidents to the pumps. Such dams are of great service in some of our collieries.

FIG. 94.



Fifth. Supposing the dam to be a pair of doors which are hung so that they can be closed to form a dam quickly in case of accident to the pumps, the hinges being one foot from bottom and top respectively, what would be the pressure in pounds upon the bottom hinges when the dam was full?

The total pressure against the dam was shown to be 11,250 pounds, which in this case would be the strain against the

hinges taken collectively. The center of pressure is a point $\frac{2}{3}$ the depth from the surface, and to find the strain on the hinges we must treat the position of the hinges on the principle of the lever. In the diagram (Fig. 94) let AB represent a dam 6 feet high, CD the hinges a foot from the bottom and top respectively, E the center of pressure or point of application of the load. Then we have a lever 4 feet long with the load applied one foot from the end, and as the length from the supports to the point of application is inversely as the relative pressure on the supports, therefore the short end gets $\frac{3}{4}$ and the long end $\frac{1}{4}$ of the load, which will in this case be equal to $11{,}250 \times 0.75 = 8437$ pounds on both hinges, or $\frac{8437}{2} = 4218.5$ pounds strain on each bottom hinge.

MISCELLANEOUS.

Ques. 19.—What method would you suggest for firing powder blasts in collieries where large quantities of explosive gas are given off?

Ans.—If an accident to, or derangement of, the ventilation cause gas to accumulate in a mine, then shot-firing should be suspended until the gas is removed. In places where the angle of dip is less than 30°, and the seam over 12 feet thick, the cross-headings for ventilation should pierce the pillars next the top, and allow the current to sweep the chambers next the roof, where gas is likely to accumulate. In such high-roofed workings, before the miner fires the shot, he should elevate the safety-lamp on a pole, or the blasting needle, and examine for gas. If any is found, it should be removed before firing. The squibs used for blasting in gaseous mines should burn without flame. If gas is issuing from a hole, it is safer to blast with a fuse or a tube; if the cartridge is carefully made and tamped, so as not to burst, then the gas cannot get into the tube to fire at the match, and is much safer than with a needle. In blasting with a needle sometimes gas blows from the needle-holes, and as the gas issues pure it extinguishes the match. If the squib is inserted through a piece of paper, the gas is deflected up behind, and the match burns on the outside. By this means shots are fired which could not be very conveniently done without it.

In some collieries in the anthracite region, the coal gives off large quantities of gas, and for the purpose of cutting the feeders and draining it off, the gangways are driven a considerable distance in advance. In some cases three gangways are driven parallel, the middle one being the main road and intake for the air, the chambers being worked from the upper gangway. The main current passes inward along the central gangway and is split into the chambers and backed to the fan. The split of air that ventilates the gangways is returned out, along the lower one, straight to the return. By this arrangement the gas drained off by the gangways does not pass into the chambers and other working places to vitiate the air. In some of these gangways the current necessary to keep them clear causes so high a velocity that a safety-lamp will not keep lighted, except with the protection of a shield; and a safety-lamp under such conditions is only useful for making an examination after the shots, or on re-entering the working face.

It frequently happens that the shots open new blowers, which the flame of the blast ignites. The one lights the other until the whole face is a mass of flame, which would soon develop into a mine fire and get beyond control. When the coal is once fairly ignited, it causes so much heat and produces so much smoke and gas, that it is dangerous and difficult to get near enough to extinguish it. To deal with such conditions, a line of pipe (generally 2½ inch wrought-iron tubing) is laid along the main gangway, from water dams on the surface. The end is fitted with a regulating globe valve, hose and nozzle. When the gas is ignited and spreads over the face so that the men cannot quench it, the hose is dragged into position. After waiting until the coal face becomes highly heated, the pressure is turned on. The water from the hose generates so much steam that by the force of the water and steam the fire is soon extinguished. Branch connections are inserted at intervals along the gangways for attaching hose to deal with fires in the chambers. By this arrangement dangerous and expensive mine fires are prevented, and at no very great expense. The pressure of the water and the promptness of its application make it the cheapest and most practical method of dealing with such conditions.

Another set of conditions where shot-firing becomes dangerous from gas is on steep angles with very thick seams, such as are found in the anthracite region. In such places where the chambers have to be worked full of coal, the face sometimes runs up ahead of the manways, and if the top part of the seam is full of slips and cracks, it may rush out and the face run up to where the air current cannot reach. The shape of the chamber makes it almost impossible to remove the gas. In such situations shots have to be fired in the vicinity of gas. One thing favorable is that in such places the coal is soft, and a good miner will not have a blow-out shot. Therefore the flame is not so apt to reach the gas. Sometimes tin cartridges are used to prevent burning paper from igniting feeders and setting fire to the loose coal. Dynamite, and giant instead of black powder, are sometimes used, as they produce less flame for the same force; but they have disadvantages which prevent their general use, even in gaseous mines. Where several shots have to be fired at once to remove pillars and start them to run, the best method is to fire with a battery, because if one shot fires in advance it may extinguish the other matches. If fuse is used, this will not occur, but the first shot may liberate gas by starting the coal to run, which may explode at the powder flame of the other shots. It has also the advantage of allowing the men to reach a safe place before firing, and the whole number of shots exploding at the same instant does better execution; the flame of the powder is extinguished before the gas is liberated. This method removes many of the objections to the usual mode of firing. All things considered, shot-firing in the vicinity of gas in the steep angles and thick seams of the anthracite region in Pennsylvania, where the coal runs, and when running liberates enormous volumes of gas, where the headings must necessarily be small and the volume of air limited, is, perhaps, under as difficult conditions for shot-firing as can be found in coal mines.

Ques. 20.—If from any cause a mine should take fire and the workings become endangered, how would you deal with it, and what methods would you adopt to extinguish it?

Ans.—Mine fires have to be dealt with according to their magnitude and the conditions surrounding them; therefore the

mode and manner of dealing with them admit of no fixed law. Extensive mine fires have generally been successfully dealt with by flooding, where flooding is practicable, either by filling the whole mine with water, or by enclosing the district in the mine where the fire is located by dams of sufficient strength to resist the pressure and flooding the enclosure. But as this method entails an enormous expense and loss of time, it is resorted to only after other methods have failed. Such dams are generally constructed by building brick arches in the narrow openings surrounding the fire, with the convex side facing the pressure.

The seats for these dams have to be cut into the solid with picks, as blasting is apt to shatter the strata and cause leakage. Some of these dams have to be placed where they cannot be reached with cars, and the material has to be carried and placed in position, in the face of great danger, requiring much time and expense. The time necessary for preparing these seats and building these dams may extend to months, and allows the fire time to extend. It is, therefore, necessary to erect temporary stoppings to exclude the air and smother the fire so as to prevent it spreading upwards to where the water could not reach it. To do this properly requires judgment and experience. If the enclosure is not complete, the fire will continue to spread; if the enclosure is tight, the gases from the fire are forced through every crevice and may endanger the workmen at the dam. If fresh air is admitted into the enclosure to carry off the gas, it may cause an explosion by mixing the carbonic oxide gas with the proper percentage of air; therefore the whole operation is attended with extreme danger. In all cases the ventilating current should be arranged so that the workmen are in the intake, and any gas from the fire will be carried off from the men. Care should also be taken that the return air cannot double back and mix with the intake.

By the time all these operations have been completed months may have elapsed, and the burning coal has raised an intense heat which reaches back into the strata. After the water is turned into the enclosure, and raised over the fire, it is, of course, extinguished, but if the water is immediately pumped out, the intense heat of the strata may rekindle the coal, and

the fire break out as fierce as before. It is, therefore, necessary to let the water cover the fire for a considerable time, to lower the temperature of the strata. Some mines have had to be re-flooded, because the water was not allowed to remain long enough over the fire. When the conditions are such that the water can only be raised a few feet above the fire, the heated water being lighter, remains on the surface in contact with the heat; and if there were any means of circulating the cold water from the bottom upwards, the cooling effect of the whole of the water could be utilized to absorb the heat from the strata and save time; but we have never seen conditions where this is practicable. The space from which a ton of coal or rock has been removed will hold over ½ ton of water; therefore, the water which is run into a mine from the surface causes a weight to be raised out of the mine again equal to over ½ of all the material previously hoisted from the space filled by the water.

Where a mine fire cannot be flooded or enclosed so as to exclude the air, then it requires different treatment. The method generally resorted to in such cases is to drill holes from the surface 6 to 10 inches in diameter, to points round the fire, which, if filled, would seal the fire air-tight; fine coal dust and silt are washed down the holes, and the openings hermetically sealed, the air excluded, and the fire gradually extinguished. If the mine is connected with the surface by only 2 or 3 shafts, incombustible gas, such as nitrogen or carbonic acid, can be produced artificially and forced through the working, which will extinguish a mine fire. It may cost more to fill the mine with gas than water, but it will cost much less to empty it. The gas may be removed by a fan in a few hours, while the time spent in pumping or hoisting the water may extend to weeks. The power expended in removing the gas is only a trifle, but the power spent in removing the water from a mine, by shafts, is equal to over ½ the whole power spent on raising the coal to the surface from the time the colliery commenced to produce coal.

Steam has often been tried as an extinguisher of mine fires, but not very successfully. The cooling surface inside is always so much in excess of the heating surface under the boilers that it can condense the steam faster than the boilers can generate it,

and steam can only act as an extinguisher where it can be confined so close around the fire as to exclude the air. And where such perfect exclusion can be effected with steam, the air can be excluded without it, and the fire would die out for want of oxygen. When fires occur in chambers of loose coal, as is often the case where the dip makes that mode of working necessary, the plan of loading and sending the burning coal out of the mine is the best and cheapest, when it can be done. Sometimes loading and moving the coal by a rush through the battery will scatter and extinguish a fire among the loose coal, but it may also spread the fire and make it worse. Torpedoes or large cartridges of dynamite exploded in the face of the chamber may have a similar effect, and both methods have at times been successful; still they cannot always be relied upon. A fire in a mine, like a fire in a city, the sooner attacked is the easier subdued. Therefore, mines should be supplied with water hose, either from the surface, or from the pump column, should have hand fire engines to run on the mine track, and fire extinguishers which can be carried up into the chambers. Some of the employees might also be drilled in the use of these means, so that the colliery would have an organized force, to act as a fire brigade, in case of accident from fire.

Ques. 21.—What is the danger from a heading approaching old abandoned workings, and what precautions would you suggest to guard against the danger?

Ans.—The danger is from water or gas, or both, and the uncertainty as to correct measurements when depending upon old surveys. In such cases two parallel headings should be driven in a direction as nearly at right angles to the supposed opening as can be ascertained. Bore-holes should be kept not less than 10 feet in advance and on each flank. One heading should be kept a few feet in advance of the other, and be used as a return for the ventilation, so that if gas is liberated the men can retire in the intake current of the other heading. The cross-headings should be driven above the bottom, so that if water is struck it cannot pass through and down the other. If the height will not admit of this, the cross-headings can be driven a little slanting upwards from the advance place, for the same purpose. The

headings should be driven as narrow as practicable, say 4 or 5 feet, and the work performed with safety-lamps. When the headings have reached as far as where known measurements render it practically safe, then all the workmen, except those employed in or connected with these headings, should be withdrawn until the drill-holes determine the facts as to danger therefrom.

Ques. 22.—If the openings of a mine are approaching the abandoned workings of another mine in which there is a head of water 100 feet vertical, how much coal would you leave as a safe barrier between the two? If it was tapped with a 2-inch drill-hole, how many gallons per minute of water would it discharge, the water maintaining the same level?

Ans.—A column of water 100 feet high would only exert a pressure at the bottom of

$$62.5 \times 100 = 6250$$

pounds per square foot, or a little over 3 tons. If the only thing to be guarded against was the water, then a pillar of, say, 15 feet would hold it, in good hard coal, across the ordinary width of a working place. If the pillar was to form a permanent barrier, then the depth from the surface would have to be considered along with other things, in regard to crushing the pillar. If the barrier was near the surface, and the pillars were to be removed, then the pillar would require to be strong enough to break off the top. The size or thickness of the pillar would have to be regulated by the character of the top and the depth from the surface. As far as the mere water is concerned, a very thin pillar would support the pressure. For a depth of, say, 300 feet, we would make the pillar 20 yards or 60 feet thick, where the pillars were to be robbed and the top broken off. If it was to hold the water only until it was tapped off, we would consider from 6 to 8 yards, according to quality and dip of coal, as a safe pillar as long as nothing was done to bring a weight, or crush, on it.

The discharge of such a drill-hole would be about as follows: The theoretical velocity due to 100 feet vertical is

$$8 \times \sqrt{100} = 80 \text{ feet per second;}$$

80 × 60 = 4800 feet per minute; 4800 × 12 = 57,600 inches per minute. The area is $2^2 \times 0.7854 = 3.1416$ square inches. The area into the velocity is 57,600 × 3.1416 = 180,956.16 cubic inches per minute. There being 277 cubic inches in a gallon,* 180,956.16 ÷ 277 = 653 gallons would be the amount of discharge. Friction in the drill-hole will reduce the theoretical quantity.

Ques. 23.—How would you ascertain the vertical depth of a shaft, having no chain or tape to measure it?

Ans.—Can only find the depth approximately with an aneroid barometer. When about to descend or ascend the shaft or slope, slide the ring around until the zero mark is opposite the pointer; then note the increase of pressure on reaching the bottom or decrease on reaching the top, as indicated by the instrument. For every $\frac{1}{10}$ of an inch allow about 90 feet; if the pointer moves one inch the vertical height will be about 900 feet; or, multiply the tenths of an inch moved by the pointer by 90, and the result is the vertical height in feet.

Ques. 24.—What precautions are necessary in using a pocket compass?

Ans.—Remove all iron or steel tools several yards from the instrument while using it. If on the mine track, hold the compass about three or four feet vertical above the centre point between the rails. These precautions are to remove all sources of attraction or to balance two opposite attractions. Keep the needle locked when carrying, to prevent damage and wear on the centre pivot and jewel of the needle from shaking or jarring incident to carrying the instrument in the pocket.

Ques. 25.—What quantity of fresh air would you consider absolutely necessary for respiration alone for a person employed at the work generally performed by laborers in the mine?

Ans.—I would consider about 30 cubic feet of pure air per hour for respiration alone; but this bears no proportion to the quantity required when other things are taken into account.

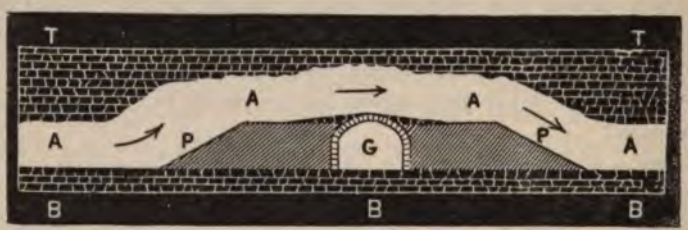
Ques. 26.—How should overcasts for the main currents in a mine be constructed?

* British Imperial gallon.

Ans.—In thick seams the overcast can be driven in the upper part of the seam, leaving a bridge of solid coal between the two airways, but in thin seams they should be constructed as shown in Fig. 95.

PP represents a pillar of coal on each side of the intake airway, *AA* is the return air course carried over it in the space blasted out of the strata *TT* above the seam. When the airway which is to cross over is driven to within 5 or 6 yards of the airway it is intended to cross, it is slanted upward on to the top of the seam, leaving the solid pillars *PP* on each side; the airway *G* is then arched over with stone or brick as shown in the figure, the arch being long enough to connect on each side of *AA*. To blast an *undercast* in the bottom strata *BB* would be

FIG. 95.



impracticable, because it would fill with water. Overcasts should be strongly built, so that when an explosion occurs something much easier repaired will give way, so the current can be quickly restored and the workings cleared of after-damp. Overcasts should not be constructed of plank or boards except for temporary purposes, and only on branch currents at unimportant places. Overcasts made of boiler iron have been proposed, and no doubt would be efficient, but the expense has prevented their introduction. We have not seen them tried, but can see no reason why they should not do for main airways where space is a consideration.

CHAPTER VIII.

BOILERS—PUMPS—PIPES.

Ques. 1.—Give your views as to the generally recognized causes which result in boiler explosions, and describe some of the kinds of boilers with which you are acquainted.

Ans.—The steam boiler, like everything else in which tremendous forces are generated, will always be apt to suddenly break through the bounds of our calculations, and apparently set all our pet theories at defiance, and when we take into account how many combinations of causes may be at work tending to develop weakness into rupture, we begin to realize the importance of investigating the subject of steam boilers. The fact that over 3000 specifications of patented designs of boilers have been recorded shows the importance of the subject and the attention it has received from inventors. Having no pretension to being a boiler expert, we do not expect, in answering this question, to do more than point out a few of the generally recognized facts concerning that rather mysterious and much abused vessel for generating steam, known as a boiler. The plain cylindrical boiler is the original type, of which the others are only more complicated improvements for economy of fuel, and as it is generally used in mining operations, we may take it for the purpose of illustrating the answer to the question.

The first thing which naturally presents itself, when considering the resistance of a boiler, is the strength of the sheets in the shell. The tensile strength of the iron is determined by the testing-machine, and the required thickness of sheet to resist the pressure is calculated from the data thus obtained. In making such calculations it should always be remembered that experiments, as a rule, are carried out under favorable conditions as to material and workmanship, and boilers made by contract, or even made to order, cannot be expected to reach the same stand-

ard as specimens in a testing-machine; therefore the margin for safety should be increased accordingly. Experiment shows that a quality of iron which in bar may have a tensile strength of from 23 to 25 tons per square inch, will, when rolled into thin sheets for boilers, have its strength reduced to 21 tons or even less, and in the transverse direction it will be even below 19 tons. The plain cylindrical boiler being weakest in the direction of the radius, should have the sheets lengthways around the boiler, and the longer the sheets the better, because it makes fewer joints in the weakest direction.

Another thing which very materially affects the strength and safety of a boiler is the workmanship. Many accidents might be traced to careless templating and punching of the holes and the use of that iniquitous instrument the "drift-pin." When the joint is lapped and the holes will not coincide, the drift-pin and the hammer sternly force them into line, and the rivets hide the forgery until the boiler leaves the shop. The fracturing of the metal may be hid by the broad heads of the rivets, and the buckling of the seams by the use of the calking tool; but as soon as the iron is subjected to expansion and contraction under severe internal pressure, the strained parts of the metal will have to do more than their due share of the elastic duty, and it is only a question of time when the metal at these points will be fatigued into a rupture ending in explosion. The force of the unfair rivets straining in opposite directions may be 50 per cent. of the ultimate strength of the iron, independent of pressure and expansion; and whenever a combination of circumstances throws a little extra strain on some of these points, a weakness is induced which only requires another strain to develop into a rupture. When a new boiler explodes without any apparent cause, the verdict is generally want of water from want of attention; and although this is doubtless often the case with boilers, still hidden flaws from imperfect workmanship often aggravate the evil and pass unnoticed.

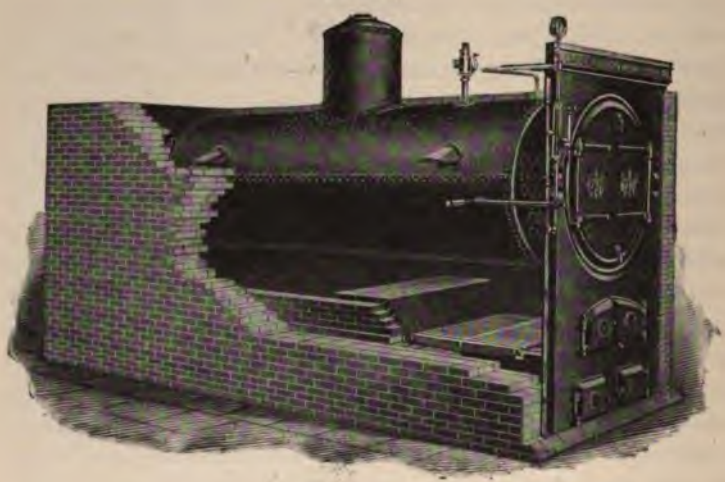
The proportions of a boiler are another factor which enters into the calculations for strength. The extent to which we can either increase the length or diameter of the plain cylindrical boiler is limited by other considerations. If we were to double

the diameter, we would need to double the thickness of the metal, and the increased thickness would interfere with the transmission of heat to the water; the outside layers of the iron would be liable to overheating, which would tend to destroy the strength of the metal from the outside inward. At the same time the double thickness would subject the inside layers to more than their due share of the double pressure, because the increased thickness prevents the outside layers from coming into play in the same proportion; therefore the extreme pressure would have a tendency to rupture the adhesive force of the iron in detail, from the inside outwards.

The length of a boiler is another thing that affects its resistance or liability to unequal strains. It is evident that the longer the tube, the greater will be the change of its length from expansion; if the middle remains stationary, the ends will travel further. If the heat was applied all around the shell, this would be of little importance; but the bottom is subjected to the direct influence of the fire, while the upper side is only heated to the temperature of the steam inside. While the under side is expanding, the upper remains almost the same length; therefore, if the ends are held fast the whole boiler must bend down in the middle. If the boiler is hung, the ends will rise to correspond to the curve, and this bending action, after a time, breaks the fibre of the iron at the edge of the lap or through the rivet holes; and when an extra bend is induced by a combination of causes, the boiler breaks across generally near the first seam behind the bridge wall, where the heated gases from the fire impinge upon the surface of the iron. This action practically sets a limit to the length that boilers can be made to work with safety; and the plan of hanging boilers at two or three places, and allowing them freedom to expand, is about all that can be done to counteract this effect in plain cylindrical boilers. This class of boiler is in general use in the anthracite region. They are made from 28 to 32 feet long and from 30 to 36 inches in diameter. Although boilers of the "Cornish" or "Lancashire" type, having the fire inside of the shell, are better economizers of heat, and as the outside shell is not subjected to direct contact with the fire, the expansion all around

is more uniform, still the boiler in use in any particular district is generally the kind which experience has proved to be best adapted to suit all the particular circumstances of that district, and the cylindrical boiler is suited in the anthracite mines for the following reasons: Their cheapness and simplicity of construction and the advantage of being easily inspected and repaired. Their small diameter enables them to resist high pressure, and their small size and light weight are convenient in moving them in a mountainous district. The water in the anthracite region is often more or less impregnated with acids

FIG. 96.



which corrode the iron and is very injurious. If lime is used to neutralize the acid, it forms a non-conducting scale on the sheets which prevents the heat from passing through the iron, causing overheating and injury. Although a plain cylindrical boiler allows a large part of the heat to escape up the stack, and is not economical in fuel, it allows a large heating surface for burning inferior grades of coal which have little market value; therefore, all things considered, they are, perhaps, the best boilers to suit all conditions. Cumbrous, complicated, and expensive internally fired boilers might produce as much steam

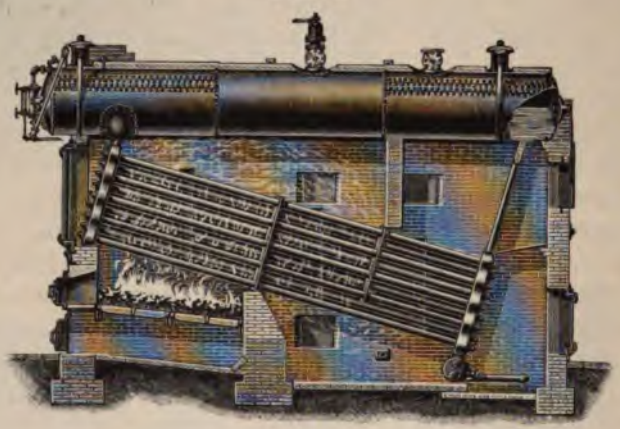
with less coal; but bad water would ruin them as quickly as others. Nevertheless, as there are other types of boilers in use about our mines, we shall describe a few of them.

The "Return Tubular" (Fig. 96) is a very common form of boiler. The furnace is placed beneath one end, as in the plain cylindrical type. The heated gases and products of combustion pass beneath to the rear, then return through a large number of tubes, passing through the boilers to the front again where they ascend the stack. This boiler is more economical of coal than the plain cylindrical type and takes up less room per horse-power. A boiler 6 feet in diameter by 18 feet long will generate 150 horse-power. It is quite cheap and at the same time simple in construction. The tubes should be regularly cleaned of soot; once a day if soft bituminous coal is used. This can be done with a steam jet. The tubes make it somewhat difficult to thoroughly clean the inside from scale and mud. A mechanical tube cleaner is manufactured which, when inserted, knocks the scale off from the outside of the tube by rapidly pounding the inside. This greatly assists cleaning, but care must be exercised in their use as they may distort and injure the tube.

The boilers we have heretofore described are what are known as shell boilers. That is to say the water is contained inside of a large closed tank or "shell." Water Tube Boilers constitute an entirely different class. In this kind the water is contained within a large number of comparatively small pipes or tubes, about which the fire circulates. The Babcock and Wilcox Boiler (Fig. 97) is a typical boiler of this class. The products of combustion pass back and forth across a great number of tubes as shown. These tubes raise from the back to the front of the boiler. As steam is generated in them it rises up grade, moving the water along with it, through the front "water-leg" into the large reservoir or top where room is allowed for the steam and water to separate. The water then returns through the back "water leg" to the lower ends of the tubes. This system of circulation plays an important part in the operation of all forms of water-tube boilers and ample room should be left in all parts to allow it. The cleaning of the inside of the boiler

is performed by opening hand holes, which are opposite each end of every tube, and inserting a cleaner. Mechanical cleaners are very convenient for this purpose. The outside of the tubes

FIG. 97.



are cleaned with a steam jet from a hose inserted into various openings in the setting.

The Stirling Boiler (Fig. 98) is another successful form of the water-tube boiler, differing in the arrangement of the tubes and reservoirs or "drums." There are fewer holes to open in cleaning it, and it is shorter from back to front than the Babcock and Wilcox Boiler.

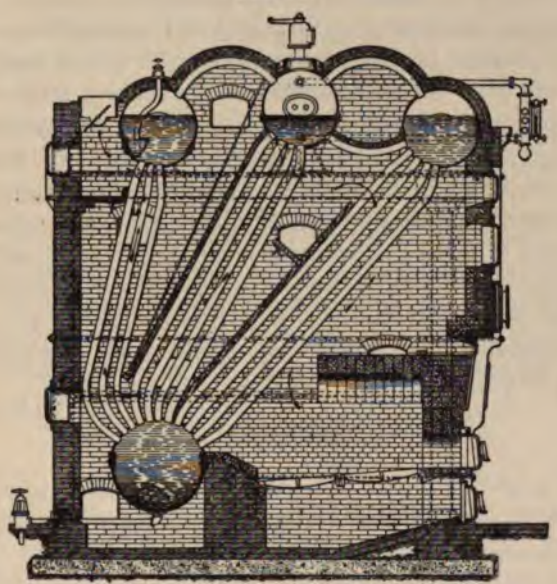
Properly built water-tube boilers are more economical in the use of coal than any other form; and as the necessity for economy increases they will no doubt come more and more into use about our coal mines. They steam quicker, and take up much less room per horse power than shell boilers and are more secure from explosion. If exploded, they cause much less damage than a shell boiler and often cause very little damage at all. For this reason they are often called "Safety Boilers." They are a costly and complicated boiler and, although they can be thoroughly cleaned, it takes more labor to clean them than it does shell boilers.

Ques. 2.—What is the resistance per square inch of a plain

cylindrical boiler of the following dimensions: Length 30 feet, diameter 30 inches, thickness of sheet ¼ inch, joints single-riveted?

Ans.—The strain to which a plain cylindrical boiler is subjected from internal pressure acts in two directions, the one tending to rupture it in the direction of the axis and the other in the direction of the radius. In order to find its strength in both directions we must know the resistance of the iron in the

FIG. 98.



THE STIRLING BOILER

boiler shell. The first factor to be considered is the tensile strength of the iron in pounds per square inch. We will assume the iron to have an ultimate tensile strength of 45,000 pounds, found by experiment in a testing-machine. Experiment shows that in boilers and similar riveted structures the strength varies with different methods of jointing, and that boilers with lap-joints, single-riveted, with punched holes, are only about 55 per cent. of the strength of the solid sheet, and as the weakest

point is the available strength of a boiler, we can only calculate on 55 per cent. of the 45,000 pounds, or

$$0.55 \times 45{,}000 = 24{,}750$$

pounds per square inch, as the ultimate resistance of the boiler structure. This may be a rather low estimate, but sufficiently correct for example. The force straining the boiler in the direction of the radius is the pressure per square inch on the number of inches in the diameter. It may on first thought seem strange that although the pressure on a boiler is so much per square inch of surface all round the circumference tending to burst it, and that the diameter is less than $\frac{1}{3}$ of the circumference, still the total pressure tending to burst the boiler is only the pressure per square inch on as many inches as make a strip one inch wide along the diameter. This is, however, demonstrated by the resolution of forces. If we imagine a ring of the boiler one inch broad like a hoop, the total pressure tending to burst it is the pressure along the diameter one inch broad, and the strength of the hoop to resist the pressure is the thickness of the hoop at both sides. The boiler shell is $\frac{1}{4}$ inch thick; therefore both sides one inch broad will be a section of $\frac{1}{2}$ square inch, and the whole resistance in the direction of the radius is equal to the strength of $\frac{1}{2}$ square inch.

$$\frac{24{,}750}{2} = 12{,}375 \text{ pounds resistance.}$$

If we now divide this strength by the diameter, it will show the pressure per square inch that the strength will resist.

$$\frac{12{,}375}{30} = 412.5,$$

being 412.5 pounds per square inch as the bursting pressure in the direction of the radius. The formula is, therefore,

$$\frac{2 \times \text{thickness} \times \text{tensile strength}}{\text{diameter}} = \text{bursting pressure, or}$$

$$\frac{\text{thickness} \times \text{tensile strength}}{\text{radius}} = \text{bursting pressure} =$$

$$\frac{24{,}750 \times 0.25}{15} = 412.5 \text{ pounds.}$$

The pressure straining the boiler in the direction of its axis is the pressure per square inch on the area of the boiler-head, and the section of the iron resisting the force is the amount of iron as measured in a cross-section of the shell. If we were to cut a ring from the boiler a foot broad and roll it up solid like a ribbon, its section would be a bolt equal to the iron in the shell. To measure its section we must multiply its circumference into its thickness. The diameter of the boiler is 30 inches; therefore its circumference is

$$30'' \times 3.1416 = 94.248$$

inches around the boiler; the thickness of the iron is one-fourth of an inch;

$$0.25 \times 94.248 = 23.562$$

square inches of iron in the cross-section of the shell. This multiplied by the strength per inch will give the whole resistance of the iron.

$$24{,}750 \text{ pounds} \times 23.562 \text{ inches} = 583{,}159.5$$

pounds as the whole strength of the boiler in the direction of its axis. This strength divided by the area of the boiler-head will show the pressure per square inch this strength will resist.

$$\text{Diameter } 30^2 = 900;\ 900 \times 0.7854 = 706.86$$

square inches as the area, and

$$\frac{583{,}159.5}{706.86} = 825$$

pounds as the bursting pressure in the direction of the axis, which is exactly double the bursting pressure in the direction of the radius. This shows us that a plain cylindrical boiler is twice as strong lengthways as it is sideways, and when a boiler bursts lengthways it is from some other cause than weakness. Changing the diameter will not change the fact of the boiler being twice as strong lengthways. To illustrate this let us change the diameter from 30 to 34 inches.

$$\frac{12{,}375 \text{ resistance}}{34 \text{ diameter}} = 363.97$$

pounds as the bursting pressure sideways;

$$3.1416 \times 34 = 106.8144,$$

the circumference of the boiler,

$$\frac{106.8144}{4} = 26.7036$$

square inches of iron in the section of the shell. 26.7036 inches $\times$ 24,750 pounds resistance = 660,914.1 total resistance.

$$34^2 = 1156;\ 1156 \times 0.7854 = 907.9224$$

square inches area in the boiler-head.

$$\frac{660,914.1}{907.9224} = 727.94$$

pounds as the bursting pressure lengthways, which is exactly double the bursting pressure sideways;

$$363.97 \times 2 = 727.94.$$

Ques. 3.—If a boiler is 30 feet long, 36 inches diameter, with the joints singly riveted and the holes punched, working at a pressure of 80 pounds, what should be the thickness of the iron in the sheets?

Ans.—The total strain on the boiler at 80 pounds will be

$$80 \times 36 = 2880 \text{ pounds;}$$

but the strength should be calculated for 5 times, or

$$2880 \times 5 = 14,400 \text{ pounds total pressure.}$$

Taking the strength of iron at 50,000 pounds per square inch and the joints at 55 per cent. of that strength, there would be a strength in the structure of about

$$50,000 \times 0.55 = 27,500 \text{ pounds.}$$

Then if we divide this pressure by the strength of the joints, it will give the thickness of the iron required in the shell.

$$\frac{14,400}{27,500} = 0.5236$$

of an inch. This is the thickness of both sides; therefore, the single thickness required is $0.5236 \div 2 = 0.2618$, or a little over $\frac{1}{4}$ inch as the required thickness.

Ques. 4.—How would you test a steam boiler to ascertain its safety?

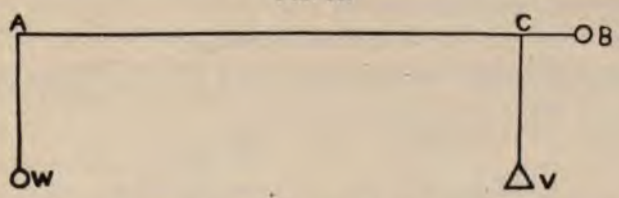
Ans.—Test it by hydraulic pressure to twice the working strain; see that the hangers are uniformly tight, and the ends

free to move by expansion. Examine the brick-work, and ascertain that the fire does not reach the shell above the water-line, and see that the safety-valve is free from sticking, that the feed and circulating pipes are free from binding, so that the expansion will not burst off the flanges, and that the gauge-cocks are free from obstruction. When the boilers are blown off for internal inspection, the surest method of ascertaining the thickness of suspected sheets is to drill a small hole through and measure the thickness. Such holes can be closed by a rivet if necessary.

Ques. 5.—If the safety-valve of a steam boiler was 4 inches in diameter, and loaded to blow off at a pressure of 80 pounds per square inch, the valve lever being 36 inches and the fulcrum 3 inches, what weight would be required at the end of the lever, the valve and lever exerting a weight of 12 pounds?

Ans.—As this is a question of leverage, let AB (Fig. 99) represent the lever, 36 inches in length, BC the fulcrum, 3 inches, V the valve, 4 inches in diameter, and W the weight at end of

FIG. 99.



valve lever. The area of the valve is $4^2 \times 0.7854 = 12.5664$ square inches; this multiplied into the steam pressure will give the total upward force against the valve, $= 12.5664 \times 80 =$ 1005.312 pounds total pressure. The valve and lever by their mere weight exert a downward pressure of 12 pounds; therefore, when this is taken off we have a net total pressure of

$$1005.312 - 12 = 993.312 \text{ pounds.}$$

The weight at the end of the long arm to balance the upward pressure can be as much less than the pressure as the distance AB is greater than BC, which gives the following proportion:—

$$36 : 3 :: 993.312 : 82.776 \text{ pounds}$$

as the weight necessary to keep down the valve to 80 pounds pressure per square inch. In all problems of this kind the light weight into the long lever is equal to the heavy weight into the short lever. In this question the leverages are to each other as 36 is to 3, or as 1 is to 12; therefore, the weight will require to be only $\frac{1}{12}$ of the total pressure, or

$$\frac{993.312}{12} = 82.776 \text{ pounds as before.}$$

Ques. 6.—If the safety-valve of a boiler is 4 inches diameter and loaded to blow off at 75 pounds pressure per square inch, the lever being 36 inches and the weight 60 pounds at the end, the weight of lever and valve being 10 pounds, what is the length of the fulcrum?

Ans.—The total pressure, minus the weight of the lever and valve, is to the weight as the whole length of the lever is to the length of the fulcrum. The area of the valve is

$$4^2 = 16; 16 \times 0.7854 = 12.5664$$

square inches. The pressure is 75 pounds per square inch, therefore the total pressure is

$$75 \times 12.5664 = 942.48$$

pounds, the weight on the lever is 60, and the weight of the lever and valve to be deducted is 10.

$$942.48 - 10 = 932.48;$$

$$932.48 : 60 :: 36 : 2.31, \text{ or } 2\tfrac{1}{3} \text{ inches}$$

as the distance from the heel of the lever to the fulcrum on the valve.

Ques. 7.—A lever 15 feet long is lifting a wagon 3 tons weight; the force on the end of the lever is equal to 200 pounds; find the position of the fulcrum, neglecting weight of lever.

Ans.—Divide the lever so that the sum of both weights is to the small weight as the whole lever is to the short end, or

$$\frac{12 \times 15 \times 200}{6000 \times 200} = 5.8,$$

or nearly 6 inches from fulcrum to short end.

Ques. 8.—What is your opinion of the practice of placing

steam boilers in coal mines, and what other methods would you adopt?

Ans.—We are not in favor of placing steam boilers or superheaters underground. The danger from fire and gas and the expense of properly securing the flues to prevent the communication of heat to the surrounding strata are grave objections to the practice. There is no doubt that where pumping engines are applied with the steam and water cylinders closely connected, or what is known as the "steam pump," if the boiler could be placed close to the steam cylinder it would be economy in fuel, because the loss from condensation would be avoided and the escaping heat from the fires together with the heat radiating from the engines could be used as a ventilating power. But the danger attending the practice is now generally recognized, few boilers, or in fact fires for any purpose, being found in our mines.

Pumping and hoisting engines underground are generally supplied with steam from boilers on the surface, and where the steam is to be used near the bottom of the shaft or slope the result is practically satisfactory. Where the steam has to be carried long distances the loss from condensation is great, and the heat caused by long lines of steam pipe is very inconvenient and is injurious to timber.

For distances over 1000 feet we would prefer to transmit power by compressed air.

Air-compressing machinery involves a greater first cost, but the increase of safety and comfort will soon pay the difference. Pipes to convey compressed air can be laid anywhere. They require no covering or expansion joints, and if a leak occurs by a car jumping the track and damaging the pipe, or a fall of top breaking it, the leak itself is no source of danger. The pipes require no traps to clear them of water, and for long distances they are more economical. All things considered, we would prefer compressed air to steam, except for short distances, as a means of transmitting power inside of coal mines.

Where hoisting has to be performed at considerable distances from the main openings we would prefer placing the boilers and engines on the surface and pass the hoisting ropes down

drill holes. This method after being tested for years gives general satisfaction.

The system of placing pumps underground and carrying steam or compressed air down one drill hole and forcing the water to the surface by another is not, as yet, in general use, although it is in operation at several collieries. When we consider that pumps are at work forcing water 1000 feet vertical in one lift without any mechanical difficulty with the valves, there can be no practical objection to the method.

Holes up to 12 and 15 inches can be rimmed to serve as a pump column. The hole is lined with a light wrought-iron tube, which allows some space round the outside of this lining. This space is run full of liquid cement which hardens into stone. If acid water should corrode the iron, the hole will still have a lining of cement. We have seen wooden pipe put in a drill hole to be used as a pump column where the water was impregnated with sulphuric acid and where iron pipe would soon be destroyed.

The system of transmitting power to underground pumps by hydraulic pressure is beginning to attract the attention of engineers, and judging from the results obtained it looks as if it may be the method of the future for long distances, the loss from friction being very little. One objection is the excessive pressure on the pipes at great depths. The pressure due to the depth in addition to the pressure transmitted through them is very great at a depth of 100 feet. With the introduction of steel instead of iron this may be practically overcome.

The method of imparting motion to the rams underground is by two small pressure columns, one to force in each direction. The columns form practically two rigid bars, the one balancing the other with no loss of water, except what may leak at the packing. This method has developed no more practical difficulties in its application than other systems of transmitting power at the same stage of their development, and it seems as if it is only a question of time when pumping from long distances in mines will take this direction.

Ques. 9.—Where boilers and machinery are placed inside of coal mines, what precautions would you recommend to guard

against fire, and how would you secure the air-course to conduct the heated gases and air to the surface?

Ans.—Where the mine is operated by shafts I would recommend that one shaft be secured with brick instead of timber to conduct the heated gases to the surface, the flues from the fire to be brick, arched with an air space between the brick work and the coal, of sufficient width for persons to pass for inspection and repairs.

Where the mine is operated by a slope I would recommend that a tunnel be driven to the nearest available thin seam and a place driven up to the surface about 15 feet wide, the coal to be carefully taken from top and bottom; then build a brick wall 3 feet from each side a foot thick, founded in a cutting at bottom and carefully pinned and jointed to the roof. The space between the walls would then form an airway with bottom and top of stone and the sides of brick. I would turn the exhaust from the engines into the spaces between the brick wall and the coal and allow a current of air to circulate freely, to carry off the heat, which would assist ventilation. At the bottom and just outside of the boiler fires I would erect a frame so that a strong dam could be placed in position by merely placing the timber which would be prepared and kept convenient. If the flues or boiler-house should take fire and endanger the mine by setting fire to the coal around the boilers or up along the flueway, then the dam could be quickly placed and water from the pumps or some other available source run into the flueway until the fire was flooded.

We would also recommend that on angles above 30° that underground pumps and engine houses be driven cross-cut into either the bottom or top rock like an ordinary tunnel. They might cost more per yard, but would need no timber and would not cut up and destroy the strength of slope or shaft pillars. They would afford solid foundation for machinery and be fireproof. They can be ventilated and kept cool by a sheet-iron brattice carried on pieces of old rail along under the roof. The danger from fire would be greatly reduced and the first cost, when timber is considered, would not be sensibly increased.

PUMPS.

Ques. 10.—If a mine makes 312.5 gallons of water per minute, and has a sump that can hold 20 hours' water, and the pump has a capacity that can empty the sump in 15 hours if the supply is cut off, how long will the pump be in taking out the water, the sump being full and the supply constant, and what will be the dimensions of the pump?

Ans.—If the supply fill the sump in 20 hours, and the pump can empty it in 15, the pump can gain 5 hours' water in every 15; and as the sump holds only 4 times 5 hours' water, the pump will gain the 20 hours' water in 4 times 15, or 60 hours. The supply amounts to $312\frac{1}{2}$ gallons; and as there are $6\frac{1}{4}$ gallons in a cubic foot, there will be

$$\frac{312.5}{6.25} = 50$$

cubic feet per minute. If the pump can gain 5 hours in 15, its capacity compared to the supply will be as 4 is to 3, which will be

$$\frac{4 \times 50}{3} = 66$$

cubic feet per minute. A steam pump 6-feet stroke, making 15 strokes per minute, would be a ram speed of

$$6 \times 15 = 90 \text{ feet},$$

a safe enough speed for large steam pumps. The required capacity in cubic feet per minute divided by the ram speed in linear feet per minute will give the area in square feet required to produce the displacement; but as pumps are measured by their diameters in inches and the ram speed by linear inches per minute, therefore the velocity would be $90 \times 12 = 1080$ linear inches, and the quantity would be 66 ft. $\times$ 12^3 = 114,048 cubic inches.

$$\frac{114{,}048}{1080} = 105.6 \text{ square inches.}$$

as the area of the ram.

$$\frac{105.6}{0.7854} = 134,$$

$$\sqrt{134} = 11.5 \text{ inches}$$

as the diameter of the pump required. In practice the pump stroke will not be quite 6 feet, but the speed can be increased. A 12-inch plunger, 6-foot stroke, will perform the work at a safe speed.

Ques. 11.—What is a Bull pump, and how does it act?

Ans.—The Bull pump cylinder is placed directly over the shaft or slope and the piston-rod connected direct to the pump-rods, which connect direct on to the ram at the bottom. The ram, the rods and the piston force the water up the column by their mere weight, and the pressure of the steam under the piston lifts them up. The engine is, therefore, single-acting, steam being only applied under the piston.

Ques. 12.—What is meant by a steam pump?

Ans.—The term "steam pump" means a pump in which the long rods connecting the ram with the piston are dispensed with. The steam cylinder and the water ram being both included in one self-contained machine, and the steam conveyed from the surface to supply the power, the displacement is produced either by a ram or a piston, single- or double-acting.

PIPES.

Ques. 13.—A pipe line in a slope is 144 square inches in area and 2000 feet long; the slope dips 1 in 10: what is the pressure in the pipe at the bottom of the slope?

Ans.—If the slope dips 1 in 10, the vertical height is

$$\frac{2000}{10} = 200 \text{ feet;}$$

this is a pressure per inch of

$$\frac{200 \times 62.5}{144} = 86.8 \text{ pounds.}$$

The diameter of a pipe whose area is 144 inches is

$$\sqrt{\frac{144}{0.7854}} = 13.54 \text{ inches.}$$

The total pressure tending to burst the pipe is, therefore,

$$13.54 \times 86.8 = 1175.272 \text{ pounds}$$

at the bottom end of the pipe. With cast-iron having a tensile

strength of 16,000 pounds per square inch, the pipe would resist this pressure if the metal was only

$$\frac{1175.272}{16{,}000} = 0.0734;$$

$$\frac{0.0734}{2} = 0.0367;$$

or about $\frac{1}{28}$ of an inch thick. Such pipes are made 20 times as thick, to stand handling and corroding by acid water.

Ques. 14.—Where mine water is strongly impregnated with acids, what kind of column pipe would you recommend?

Ans.—Cast-iron column pipes lined with thin pine laths. The laths are narrow strips with the sides planed off at a level to suit the radius. They are put in dry and fitted close without being tight; if they are too tight, the water swelling the timber may *buckle* them out; if slack, they swell tight, but not so as to cause buckling. The ends are dressed off to meet without buckling, when the flange-bolts are screwed tight.

Ques. 15.—What is the principle of the siphon, and to what height can it be applied to raise water? Explain how you would have it arranged for charging and starting the flow.

Ans.—The principle of the siphon is atmospheric pressure. When the tube is full, the water from the highest point runs down toward the discharge end, and the pressure on the surface of the water at the supply end forces the column up to fill the vacuum as long as the vertical height from the surface of the water at the supply to the highest point does not exceed the height to which the pressure of the air will force water, as in a pump, which must be less than 34 feet at sea level. The same pressure is preventing the water from flowing out at the lower end; therefore, the velocity of the flow depends on the difference of level between the surface of the water at the supply and the delivery ends of the tube. The exact height to which a siphon will raise water of course depends on the elevation of the place where it is put and the density of the liquid it is acting upon. A siphon will lift water higher in the bottom of a shaft below tide than on the top of a high mountain. It will lift water more than 30 feet, and mercury less than 30 inches.

In placing a siphon for practical use I would immerse the discharge end of the tube, covering the orifice with a few inches of water. Both ends of the tube should be fitted with a globe valve and the summit with a branch pipe and globe valve. To charge and start the flow, close the valves at both ends, fill the siphon tube by means of the branch pipe; when full close the branch pipe and open the valve at the supply end, and lastly open the valve at the discharge end, when the flow will commence. The discharge can be regulated by the valve to prevent it running off faster than the supply and breaking the vacuum, thus obviating the necessity of recharging and starting.

Ques. 16.—Explain the formulæ for the flow of water in pipes.

Ans.

Let D = Diam. of pipe in inches.

Let L = Length of pipe in ft.

Let H = Total head in ft. = height of surface of reservoir above the discharge end of pipe.

Let h′ = Head in feet necessary to generate the velocity in the pipe.

Let h = Head in ft. necessary to overcome the friction of the pipe.

Let v = The velocity of the water in the pipe in ft. per second.

Let Q = The quantity of water discharged in cubic ft. per second.

Let g = The acceleration of gravity in ft. per second = 32.

In entering the pipe the water must be raised to the velocity with which it flows. The head necessary to do this is theoretically equal to the height through which a body must fall to reach that velocity, or:

$$h' = \frac{v^2}{2g}.$$

Practically it takes about $1\frac{1}{2}$ times this head, hence:

$$h' = \frac{1.5\, v^2}{2g}.$$

But $v = \frac{Q}{\text{area}} = \frac{Q \times 4 \times 144}{3.142 \times D^2}$;

therefore by substitution

$$h' = \frac{1.5 \times Q^2 \times 4^2 \times 144^2}{2 \times 32 \times 3.142^2 \times D^4},$$

which reduces to:

$$h' = \frac{800\ Q^2}{D^4}. \qquad (1)$$

By solving we get:

$$Q = .035\ D^2 \sqrt{h'} \qquad (2)$$

$$D = 5.3 \sqrt{\frac{Q}{\sqrt{h'}}}. \qquad (3)$$

Where the length of the pipe is not greater than twice the diameter (or 1 ft. long for a 6 in. pipe), the friction of the water against the sides of the pipe may be neglected and h' is practically equal to the total head H. We may therefore use formulæ (1), (2) and (3) for pipes of such lengths.

But if the pipe were longer than twice the diameter we would have to take the friction into account. The force necessary to overcome the friction against the sides of the pipe is proportional to the inside "rubbing surface" of the pipe and to the square of the velocity. This force divided by the sectional area of the pipe gives the pressure per square inch necessary to overcome the friction. As the head is proportional to this pressure we can write:

$$h \text{ is proportional to } \frac{\text{surface} \times v^2}{\text{area}}.$$

But the surface is proportional to the length L and the diameter D; therefore it is proportional to L D.

The velocity is proportional to the quantity Q and inversely proportional to the area; therefore it is proportional to $\frac{Q}{\text{area}}$. Substituting these in our proportional equation, we get:

$$h \text{ is proportioned to } \frac{L\,D\,Q^2}{\text{area}^3}.$$

But because the area is proportional to the diameter D squared we can write:

$$h \text{ is proportional to } \frac{L\,D\,Q^2}{D^6} = \frac{L\,Q^2}{D^5}.$$

Now when we say h is proportional to the above quantity, we mean that it is equal to some number multiplied by that quantity. By experiment this number has been found to be 200. So we can write:

$$h = 200 \frac{LQ^2}{D^5}. \qquad (4)$$

By solving for Q and D we get:

$$Q = .07\sqrt{\frac{D^5\, h}{L}}, \qquad (5)$$

$$D = 2.9\sqrt[5]{\frac{L\, Q^2}{h}}. \qquad (6)$$

If the pipe were a very long one, say 1000 times the diameter, or about 100 ft. long for every inch in diameter, then nearly all of the total head H would be absorbed by friction or h would practically equal H. The above formulæ (4), (5) and (6) are often used for such cases.

To combine these two sets of formulæ into a single set suitable for all pipes we have:

$$H = h' + h.$$

By substitution

$$H = \frac{800\, Q^2}{D^4} + \frac{200\, L\, Q^2}{D^5}$$

which reduces to

$$H = 200\, Q^2\left(\frac{L + 4D}{D^5}\right) \qquad (7)$$

From which the value of Q in terms of the diameter and total head is deduced:

$$Q = .07\sqrt{\frac{D^5\, H}{L + 4D}}. \qquad (8)$$

This question cannot be conveniently solved by any direct method for the value of D. On this account and because they are more complicated, equations (7) and (8) are not used except when it is necessary to calculate for both velocity and friction heads.

Ques. 17.—How much head will it take to deliver 800 U. S. gallons per minute through a 6″ pipe line 1000 ft. long?

Ans.—800 gals. $= 800 \times 231$ cu. inches per minute. This equals $\frac{800 \times 231}{60 \times 1728}$ cubic ft. per second $= 1.78 = Q$.

As the pipe is more than 1000 times the diameter we can use formula (4).

$$h = H = 200 \frac{LQ^2}{D^5}$$

$$= 200 \frac{1000 \times 1.78^2}{6^5}$$

$$= \frac{200{,}000 \times 3.17}{7800}$$

$$= 81 \text{ ft.}$$

Ques. 18.—How much water will a 4″ pipe 300 ft. long deliver when the height between the top of the water in the tank and the outer end of the pipe is 50 ft.?

Ans.—This problem could be solved by the use of the friction formula because its length is about 1000 times its diameter, but for illustration we will solve it both ways. Formula (8) is:

$$Q = .07 \sqrt{\frac{4^5 \times 50}{300 + 4 \times 4}} = .89$$

cubic ft. per second. And

$$\frac{.89 \times 60 \times 1728}{231} = 398$$

U. S. gallons per minute.

If we had used formula (5) we would have:

$$Q = .07 \sqrt{\frac{4^5 \times 50}{.300}} = .92$$

cubic feet per second.

This is quite near enough for practical calculation.

Ques. 19.—What diameter must a pipe be to deliver 3 cu. ft. of water per second when the head is 40 ft. and the pipe is 100 ft. long?

Ans.—There is no convenient and direct way of solving this question. We might use formula (6) if we thought the diameter was going to come out less than $\frac{1}{1000}$ of the length, but as

the length is short we had better use the complete equation No. (8),

$$Q = .07 \sqrt{\frac{D^5 H}{L + 4\,D}}.$$

The only way to get the value of D from this is by trial thus:

$$3 = .07 \sqrt{\frac{40\,D^5}{100 + 4\,D}},$$

or by squaring:

$$9 = \frac{.005 \times 40 \times D^5}{100 + 4\,D}$$

$$9 = \frac{.2\,D^5}{100 + 4\,D}$$

If D = 3″ the above fraction becomes $\frac{.2 \times 3^5}{100 + 12} = .43.$

Hence we must try a larger diameter.

When D = 5″; then $\frac{.2 \times 5^5}{100 + 20} = 5.2$

when D = 6″; then $\frac{.2 \times 6^5}{100 + 24} = 12.5$

Because the fraction ought to equal 9 we conclude that our pipe should be between 5″ and 6″ in diameter. As 6″ is the next larger size we will make it a 6″ pipe.

Then:

$$Q = .07 \sqrt{\frac{6^5 \times 40}{100 + 24}} = 3.5$$

which shows it is a little larger than required.

In problems of this kind if the diameter is found to be less than 2″ it should be increased by about ¼ of the diameter. This is not because of safety but because the formula gives low results for such small pipes, and because they are more liable to become stopped up.

For the same reason the discharge given for pipes under 2″ is probably a little high and the head necessary for a given discharge is a little low.

CHAPTER IX.

HOISTING—HAULAGE.

Ques. 1.—If an engine of 60 horse-power can hoist with a cage speed of 600 feet per minute, what load will it hoist in a shaft, and what load will it haul up an inclined plane having a gradient of 1 in 3?

Ans.—The foot-pounds or units of work in a 60 horse-power engine are equal to

$$33{,}000 \times 60 = 1{,}980{,}000;$$

which is equal to 1,980,000 pounds raised 1 foot in 1 minute, or 1 pound raised 1,980,000 feet in 1 minute. Therefore if the engine moves the load at 600 feet per minute, if we divide the pounds raised 1 foot in a minute, or the feet that 1 pound is raised in a minute by 600, the result will be the pounds raised 600 feet in a minute.

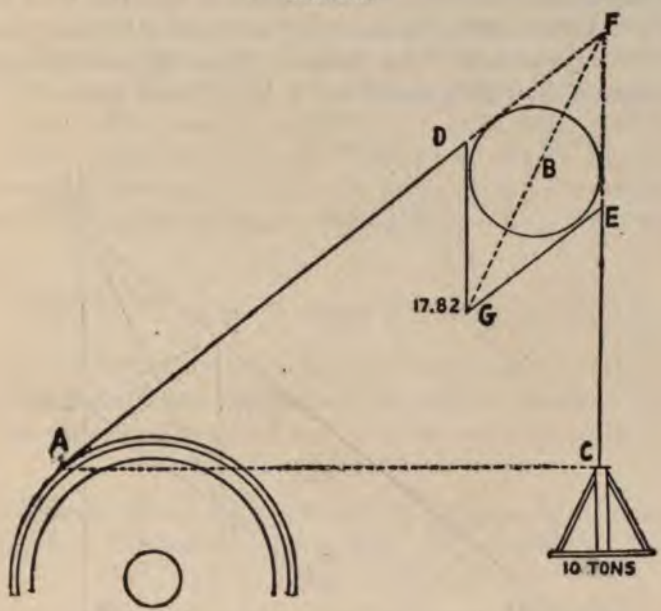
$$\frac{1{,}980{,}000}{600} = 3300$$

pounds as the load which the engine can hoist. With a single engine the cross-head must be near the middle of the guides at the lift. If the crank is near the centre at the lift, the engine may not be able to start the load. This has to be provided for in the adjustment of the ropes. In hauling up a plane dipping 1 in 3 the load will travel 3 feet for 1 foot vertical; therefore it will haul 3 times the load less what extra friction may be caused by the bearing rollers and the extra length of rope for the same vertical depth.

Ques. 2.—How would you determine the strain upon the sheave over a hoisting shaft, when the weight of cage, car, load, and rope amounts to 10 tons, the rope leaving the drum at an angle of 36°, and what should be the diameter of the sheave shaft, the bearings being 6 feet apart, and the shaft of wrought-iron?

Ans.—The strain upon the shaft is the resultant of the strain caused by the weight being lifted and the strain caused by the tension of the rope on the opposite side of the sheave which is lifting it, and is determined by the parallelogram of forces either

Fig. 100.



by construction or calculation. To illustrate the method of construction in Fig. 100, let

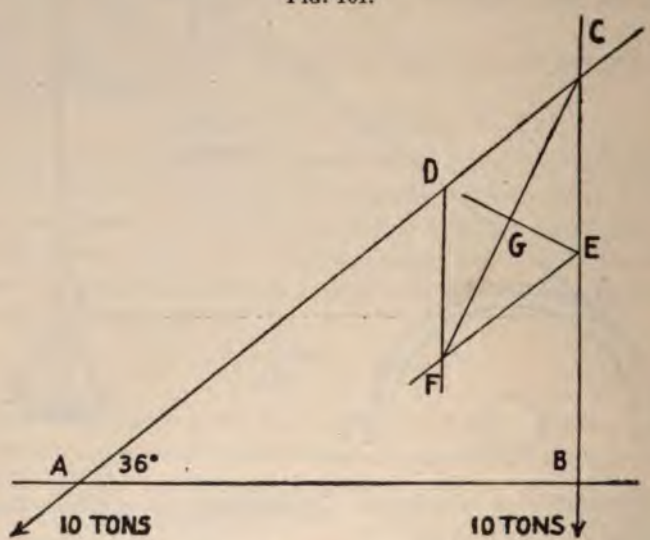
A	represent	the	drum.
C	"	"	load.
AD	"	"	rope to the sheave.
EC	"	"	rope in the shaft.
B	"	"	sheave.

Produce the lines *AD* and *CE* to *F;* from *F* along both lines lay off as many equal parts as there are tons in the load; from the end of the last divisions at *D* and *E* draw *EG* parallel to *FD* and *DG* parallel to *FE;* from their point of intersection *G* to *F* draw the diagonal *FG*, divide the line *FG* by the same

scale of equal parts, and the number of divisions on *FG* will represent the tons of strain on the sheave. To find the strain by this method requires very careful measurement on a large scale, so that fractional parts of the angles and lines are of sufficient magnitude to be accurately measured.

To determine the strain by calculation in Fig. 101 let *A* be a strain of 10 tons at the drum, and *B* a weight of 10 tons on the rope, *C* the position of the sheave. Make *CD* and *CE* equal, and from *E* draw *EF* parallel to *CA*, join *CF* and *ED*.

FIG. 101.



Because *CD* and *CE* are equal and *DF* and *EF* are equal and parallel to them, the four sides of the quadrilateral *CDFE* are equal. Now because the weight of 10 tons at *B* is balanced by a force of 10 tons pulling at *A*, and because *CD* and *CE* represent these forces in magnitude and direction, therefore, by the parallelogram of forces, the diagonal *CF* will represent in magnitude and direction the resultant of these forces acting on *C*, which is the strain upon the sheave. We must, therefore, find the value of *CF*.

Because the angle at *B* of the triangle *ABC* is a right angle,

the angles at A and C are together equal to a right angle, and as the angle at A is 36° the angle at C is equal to 90° — 36° = 54°, and because the diagonal CF bisects the angle DCE, the angle ECG is equal to $\frac{1}{2}$ of 54°, which is 27°. Because the angle EGC is a right angle we have the right-angled triangle EGC with the angle at C equal to 27°, and the hypothenuse CE equal to 10; therefore, if we multiply 10 by the cosine of 27° it will give the length of the cosine CG. The cosine of 27° is 0.89101, and 0.89101 × 10 = 8.9101, the length of CG; because CG is $\frac{1}{2}$ of CF it is equal to 8.9101 × 2 = 17.8202; therefore, the strain upon the sheave is 17.82 tons. Hence, the strain upon a sheave is equal to twice the cosine of half the contained angle ACB neglecting friction and rigidity of rope. The formula is

$$\frac{\text{Contained angle}}{2}\ \text{cosine} \times \text{weight} \times 2 = \text{strain on sheave.}$$

The value of the strain changes with every angle. If the rope was horizontal from the drum to the sheave, the strain would be the diagonal of a square, and its value would be equal to

$$\sqrt{\text{Load}^2 \times 2}.$$

The diameter of the shaft is found by the formula for strength of beams. The load on the sheave is 17.82 tons, or

$$17.82 \times 2000 = 35{,}610$$

pounds; but to insure rigidity and give ample strength to resist sudden jars, we may calculate the shaft for 10 times the load. This would be

$$35{,}640 \times 10 = 356{,}400$$

pounds. The formula generally given to find the diameter of round shafts, loaded in the middle, is deduced from two facts: first, that a round bar of wrought-iron one inch in diameter 12 inches long, supported at both ends and loaded at the centre, will break with a weight of about 2000 pounds; and secondly, that the strength of such bars varies directly as the cube of the diameter and inversely as their length, therefore the formula is

$$\frac{\text{Length in feet} \times \text{weight}}{\text{Constant} = 2000} = \text{cube of diameter.}$$

Applying this to our question we have

$$\frac{6 \times 356{,}400}{2000} = 1069.2$$

inches as the cube of the diameter, and

$$\sqrt[3]{1069.2} = 10.2 \text{ inches, or}$$

$2000 \times 1 : 356{,}400 \times 6 :: 1^3 : 1069.2$, $\sqrt[3]{1069.2} = 10.2$ inches. This is the strength calculated for 10 times the load, and would be amply sufficient to insure rigidity and safety.

Ques. 3.—How would you test the strength of a wire rope?

Ans.—Cut a sample from the rope, put a socket on each end, and find its ultimate tensile strength with a testing-machine.

Ques. 4. On a slope 500 yards long, dipping at an angle of 45°, what would be the thickness of an iron wire rope to hoist two cars, each 3 tons weight?

Ans.—According to the manufacturer's tables, a rope 1 inch in diameter is allowed a working-load of 3 tons, and 5 times, or 15 tons, as the breaking-strain. Now as the strength of ropes will vary as the material in their section, all other things being equal—which, with a round rope, is in proportion to the square of the diameter—we have the following formula to determine the relative strength of ropes:

First. To find the diameter when the load is given—

$$3 \text{ tons} : 1 \text{ inch}^2 :: \text{given load} : \text{required diameter}^2,$$

and the square root is the diameter, or

$$\sqrt{\frac{\text{Load}}{3}} = \text{diameter.}$$

Second. To find the load where the diameter is given—

$$1 \text{ inch}^2 : \text{diameter}^2 :: 3 \text{ tons} : \text{required load;}$$

or,

$$\text{Diameter}^2 \times 3 = \text{load.}$$

For steel ropes, the same formula, with 5 instead of 3 tons as the unit and 5 times the working-load as the breaking-strain, as with iron.

The relative weight of ropes per foot is also in proportion to the material in their section; and as the section varies as the

square of the diameter, the same formula will find the weight per foot as well as the strength, when the weight per foot of a 1-inch rope is known.

Example.—What is the weight of a 2-inch iron wire rope 1000 feet long?

Ans.—The weight of a 1-inch rope is given at 1.58 pounds per foot; but to be easily remembered, we will call it 1.60, which is erring on the safe side when making allowance for weight of rope; then diameter^2 $\times$ weight of 1-inch rope $= 2^2 \times 1.6 = 6.4$ pounds per foot, and $6.4 \times 1000 = 6400$ pounds, or $3\frac{1}{5}$ tons. When the weight is given, to find the diameter first divide the weight by the length to find the weight per foot; then divide the weight per foot by 1.6 and take out the square root, or

$$\sqrt[2]{\frac{\text{Weight per foot}}{1.6}} = \text{diameter.}$$

Steel ropes are about the same weight. In the first question the load is given and the diameter required. Two cars 3 tons each $= 6$ tons $= 12{,}000$ pounds. To find the gravity due to 6 tons on an angle of 45°, we must multiply the weight by the sine of 45°:

$$0.70711 \times 6 = 4.24266,$$

or nearly $4\frac{1}{4}$ tons as the strain on that pitch. A rope to work with a load of $4\frac{1}{4}$ tons should have a diameter of

$$\sqrt{\frac{4.24266}{3}} = 1.189 \text{ inches.}$$

or nearly $1\frac{1}{4}$ inches diameter. Now the rope has to lift its own weight in addition to the load; and, according to the tables, a $1\frac{1}{4}$-inch rope weighs $2\frac{1}{2}$ pounds per foot.

500 yards $\times$ 3 $=$ 1500 feet; $1500 \times 2.5 = 3750$ pounds. Add for friction on rollers, say, 250 pounds more, and the total load to be lifted from the bottom is $3750 + 250 + 12{,}000 = 16{,}000$ $=$ 8 tons total load.

$$8 \text{ tons} \times \text{sine of } 45° = 5.656 \text{ tons;}$$

$$\sqrt{\frac{5.656}{3}} = 1.37 \text{ inches diameter.}$$

We only add the weight of a rope to lift 6 instead of 8 tons; therefore, we should have added the weight of a 1.37-inch rope; but if we increase the diameter to a $1\frac{1}{2}$-inch rope, it will be about the required size for the work, and, in order to make ample allowance for friction and speed, would say a rope $1\frac{3}{4}$ inches as the required thickness.

Ques. 5.—What is the working load of a 2-inch steel rope on a slope 500 yards deep, pitching 60°?

Ans.— $2^2 \times 5 = 20$ tons

as the working-load, which, on an angle of 60°, is equal to 20 tons divided by the sine of 60°, or

$$\frac{20}{0.86602} = 23.09 \text{ tons.}$$

But as the rope has to lift its own weight at the bottom, it must be deducted from the load. By the maker's table, the weight of a 2-inch steel rope is 6.3 pounds per foot; the weight of the rope would, therefore, be

$$500 \text{ yards} \times 3 = 1500 \text{ feet}; \quad 1500 \times 6.3 = 9450 \text{ pounds};$$

by allowing, say, 550 pounds for friction on rollers to be added to the weight of the rope we would have

$$9450 + 550 = 10{,}000 \text{ pounds} = 5 \text{ tons},$$

leaving a net working-load for the rope of

$$23 - 5 = 18 \text{ tons.}$$

Ques. 6.—What is the strain on a rope caused by a load of 6 tons on a slope pitching 25°, and what is the strain on a slope pitching 60°?

Ans.—In all cases of this kind we multiply the load by the sine of the angle of dip. On the slope pitching 25°, the strain is

$$6 \times 0.42262 = 2.53572, \text{ or a little over } 2\tfrac{1}{2} \text{ tons.}$$

On the slope pitching 60° the strain is equal to

$$6 \times 0.86603 = 5.19618, \text{ or nearly } 5\tfrac{1}{5} \text{ tons.}$$

Ques. 7.—What weight would you consider a safe working-load for a hemp rope 2 inches in diameter?

Ans.—The ultimate strength of good hemp rope is usually considered to be about 6400 pounds per square inch of section.

As a rule, the nominal size of hemp ropes refers to the circumference, but as we generally use the diameter when speaking of wire rope, we will reduce the nominal size to suit the formula already given and find the unit for hemp rope as follows:—

If the ultimate strength of a hemp rope is 6400 pounds per square inch of section, then the strength of a round rope 1 inch in diameter will be

$$1 : 0.7854 :: 6400 : 5027 \text{ pounds.}$$

If we take $\frac{1}{5}$ of the breaking-strain as the working-load, it will be about

$$\frac{5027}{5} = 1005 \text{ pounds,}$$

or $\frac{1}{2}$ ton, and the formula for hemp rope becomes

$$\sqrt{\frac{\text{Load}}{0.5}} = \text{diameter.}$$

$$\text{Diameter}^2 \times 0.5 = \text{load.}$$

This applied to the question would give $2^2 \times 0.5 = 2$ tons as the working-load, and $5 \times 2 = 10$ tons as the load required to break the rope.

Ques. 8.—Suppose you are required to lower large column pipes in a shaft, each of which weighs from 1200 to 1500 pounds, what size of hemp rope will you require?

Ans.—Assuming the greatest load at 1500 lbs., or $\frac{3}{4}$ of a ton, the rope would require to be

$$\sqrt{\frac{0.75}{0.5}} = 1.23 \text{ inches;}$$

therefore a good $1\frac{1}{4}$ inch hemp rope would do the work and require $3\frac{3}{4}$ tons to break it if the depth did not exceed 100 feet.

The following formula is sometimes given to find the strength of hemp rope:

$$\text{Breaking weight in tons} = \frac{\text{circumference}^2}{4}$$

$$\text{Working loads in tons} = \frac{\text{circumference}^2}{10}$$

Ques. 9.—Is the strain due to the load, the weight of the rope, and the friction, all the strain that a hoisting rope is subjected to?

Ans.—The strain upon a hoisting rope is not constant even with a constant load. When the engine starts the load it has to overcome its inertia and the power necessary to raise the speed has to be transmitted through the rope. At the instant of changing the load from rest to motion, the strain upon the rope is according to the suddenness of the lift and the amount of velocity imparted to the load in a given time. The principle is not so apparent with the slow speed of hoisting usual in slopes and shallow shafts, but when we come to first motion engines and large drums, such as are now being introduced at our new collieries, this principle of speed becomes a matter of great importance when calculating the strain upon the hoisting rope.

At some of the collieries in England a cage speed of 3000 feet per minute, or 50 feet per second, is attained at a distance of 900 feet from the bottom of a shaft 2145 feet deep with a load of 7 tons. When we consider the enormous amount of work stored up in a weight of 7 tons at that velocity, and that the necessary power has to be transmitted through the rope, we begin to realize the strain to which the rope must be subjected while getting up speed. For the purpose of illustration let us take the above example. Suppose the rope has overcome the inertia of the load, and at 20 feet up the shaft the speed is 10 feet per second. The work thus stored up in the load as speed at that instant is found by the formula for gravity.

A body to acquire a velocity of 10 feet per second would require to fall a height equal to

$$\frac{v^2}{2g} = \frac{10^2}{64.4} = 1.55 \text{ feet};$$

therefore the work stored up in the load at that speed would lift it through a height of 1.55 feet, and as the foot pounds is the load into the height through which it is lifted, this is equal to

$$7 \text{ tons} = 14{,}000 \text{ pounds}; \quad 14{,}000 \times 1.55 = 21{,}700$$

foot pounds as represented by the load at that speed.

Let us apply the same law to the load when it has reached its full speed of 50 feet per second at 900 feet up the shaft.

$$\frac{50^2}{64.4} = 38.82 \text{ feet nearly};$$

therefore the speed now stored up in the load would lift it through a height of nearly 39 feet.

$$38.82 \times 14{,}000 = 543{,}480$$

foot pounds, as represented by the load at that speed.

The force impressed upon the load which has changed its speed from being able to lift itself through $1\frac{1}{2}$ feet to lifting itself through 39 feet, has been transmitted through the rope.

When we consider that the friction on the guides and shocks from the vibration will increase in intensity as the square of the speed, this and the strain of starting the load and getting up speed all combined cause a strain on the rope which we cannot afford to neglect when calculating the required strength of rope.

Ques. 10.—The weight of a cage, car and load is $8\frac{1}{2}$ tons, it is connected to the rope by two chains, what should be the diameter of the iron in the links?

Ans.—Experiment has shown that a single-link chain of good iron carefully welded, made of round iron one inch in diameter, has a safe working strength of about 6 tons, and that the strength of chains varies as the square of the diameter of the iron of which the links are made or as the section of the iron in the chain; therefore, in this question,

$$6 \text{ tons} : 8\tfrac{1}{2} \text{ tons} :: 1 \text{ inch}^2 : 1.4166 = \text{inches}^2, \text{ and}$$

$\sqrt{1.4166} = 1.19$ inches as the size of a chain to carry the whole load; but, owing to oscillation and vibration, nearly all the load may be sometimes thrown on one chain; therefore, to allow an ample margin for safety, both chains should be made of iron $1\frac{1}{4}$ inch in diameter. The formula will therefore be:—

$$\sqrt{\frac{\text{Load}}{6}} = \text{diameter of iron in link.}$$

Ques. 11.—A column of pipes is to be raised by a chain and block; the calculated weight of the column is 36 tons; what size of chain should be used?

Ans.—There being two chains from the block, each will have a load of $36 \div 2 = 18$ tons.

$$\frac{18}{6} = 3, \text{ and } \sqrt{3} = 1.73 \text{ inches;}$$

allowing for friction, the chain should be made of iron $1\frac{3}{4}$ inches in diameter, or a $1\frac{3}{4}$-inch chain. When the chain is given, the working load is found by

$$\text{diameter}^2 \times 6 = \text{working load in tons,}$$

the ultimate strength being about four times the working load. Very large or very small chains are rather weaker than this rule, the large sizes being more apt to be made of inferior iron, and the small sizes more apt to have imperfect welding.

Another simple rule to find the approximate safe working load of a chain is to square the number of one-eighth of an inch in the iron and strike off the last figure as a decimal. In an inch chain there are 8 eights; then $8^2 = 64$, and $6.4 = 6$ and $\frac{4}{10}$ tons = the safe load.

Ques. 12.—Give your views as to the causes at work which result in chains breaking with strains much less than their calculated strength, and what precautions would you suggest in the treatment of chains on which the safety of life and property depends?

Ans.—The exact cause which results in the sudden breaking of a chain cannot be easily traced. It may be due to inferior material, imperfect workmanship, or violent usage.

Standard chains are tested up to about one-half of their ultimate calculated strength, or about double the ultimate working load; but after they have been working a short time some link may begin to stretch, especially in chains used for severe strains. If one link, either from imperfection in the iron or welding or from want of uniformity in its section, should be originally weaker than the others, when a heavy strain or sudden jerk affects the chain the weak link will have to perform more than its due share of elastic duty, and if not cut out will ultimately break with a load much less than the calculated working strain. Chains which are subjected to sudden blows on hard surfaces, such as at top and bottom of slopes and planes, where the chain is thrown off and falls on rails or stones, such violent vibration crystallizes the iron, until it will break by a stroke, like glass. A chain with links in this condition is dangerous, and should be annealed or heated to a dull red and allowed to cool slowly. This heating will restore the strength by putting the material of

the links back into its original state of equilibrium, although every time a chain is heated it loses some of its original strength. Another and perhaps the most common cause of chains suddenly giving way is from imperfect welding. This can only be guarded against by close inspection by a blacksmith or some other experienced person. We would, therefore, suggest that chains on shafts, slopes, and planes be carefully inspected weekly, and those which are thrown on rails and violently jarred while at work be annealed at least once a year.

Ques. 13.—What points in a hoisting plant would you first examine when inspecting in the interest of safety?

Ans.—(1) The ropes, (2) the capping or sockets, (3) the chains, (4) the indicators, (5) the brake, (6) the general details of the engine.

Ques. 14.—How do you find the horse-power of a steam-engine?

Ans.—The following formula is generally used to find the power:—

$$\frac{A\ P\ S\ 2\ R}{33{,}000} = HP.$$

Where A = area of piston in square inches.
" P = average pressure per square inch.
" S = length of stroke in feet.
" R = revolutions per minute.
" HP = horse-power.

Which means: Square the diameter of the cylinder in inches; multiply it by 0.7854, which gives the area; this multiplied by the pressure per square inch gives the total pressure or load upon the piston; this multiplied by twice the length of stroke gives the distance in feet which the piston travels with the load at each revolution; and this again multiplied by the number of revolutions per minute gives the total distance in feet traveled by the piston with the load in one minute, which result is the foot-pounds or units of work; and this divided by 33,000 reduces it to horse-power, the same as reducing pounds to tons.

Ques. 15.—What is the available power of a pair of hoisting engines with 24-inch cylinder, 5-feet stroke, making 40 revolu-

tions per minute, the gauges at the boilers indicating a pressure of 75 pounds?

Ans.—Assuming one-third of the pressure to be spent in overcoming friction, and that the difference of pressure between the boilers and the pistons would leave an effective average pressure of 50 pounds; then

$$24^2 \times 0.7854 = 452.3904 \text{ square inches}$$

as the area of the piston;

$$\frac{452.40 \times 50 \times 5 \times 2 \times 40}{33{,}000} = 274.18 \text{ HP}$$

as the power of one engine; and

$$274.18 \times 2 = 548.36 \text{ HP}$$

as the power of the pair at that speed and pressure.

Ques. 16.—What type of engines would you prefer for hoisting at collieries?

Ans.—Would prefer high-pressure, horizontal double, first motion engines with balance cranks; and where the depth was over 500 feet would prefer conical drums, and have the engines fitted with balance-valves worked by drag-cranks from the main crank-pins; the cylinders to be of such size as to develop the required power at not more than 500 feet of piston speed per minute.

Ques. 17.—A shaft is 1600 feet deep; the ropes are steel, $1\frac{3}{4}$ inches in diameter; the cages are double-decked, built of steel, and weigh $1\frac{1}{2}$ tons; the empty cars weigh 1 ton each and carry 3 tons of coal; the engines are first motion with cone drum, the small diameter of the cone is 14 feet: what should be the large diameter?

Ans.—In order to solve this problem we must first find the weight of the rope. A steel rope $1\frac{3}{4}$ inches in diameter, with wire centre, will weigh about 5.66 pounds per foot;

$$\frac{5.66 \times 1600}{2000} = 4.5 \text{ tons, nearly;}$$

therefore the load to be started by the engines will be about as follows:—

Weight of 2 car-loads of coal,	6 tons.
" of 2 cars,	2 "
" of cage,	$1\frac{1}{2}$ "
" of rope,	$4\frac{1}{2}$ "
Total load,	14 tons.

When the load reaches the top it has lost the weight of the rope and is only

$$14 - 4.5 = 9.5 \text{ tons;}$$

then if the engine can lift 14 tons on a 14-foot lever, it will lift $9\frac{1}{2}$ tons on a 20.7-foot lever, because .

$$\frac{14 \times 14}{9.5} = 20.63 + \text{ feet;}$$

therefore the large end of the cone should be 20 feet 8 + inches, if we neglect the balance from the other rope. Let us now examine the effect of the rope, cage, and empty cars on the other side. At the lift we have 14 tons on a 14-foot diameter and a back balance of $3\frac{1}{2}$ tons on a 20.63-foot diameter; $3\frac{1}{2}$ on 20.63 is equal to 5.16 on 14, because

$$\frac{3.5 \times 20.63}{14} = 5.16,$$

and

$$14 - 5.16 = 8.84$$

as the unbalanced load at the lift.

When the back balance reaches the bottom it is 8 tons on a 14-foot diameter, and 8 on 14 is equal to 5.42 on 20.63, because

$$\frac{8 \times 14}{20.63} = 5.42,$$

and the load at top of

$$9.5 - 5.42 = 4.08$$

as the unbalanced load at the top.

But an unbalanced load of 8.84 on a 14-foot drum would be equal to 5.99 tons on a 20.63-foot drum; and the actual unbalanced load being only 4.08 tons, we at once see that the large diameter should be more than 20.63 feet to be theoretically correct.

The following formula is given to solve the problem:

$$D = \frac{d \times (W + 2C + 2R)}{W + 2C}.$$

Where W = weight of coal in cars.
Where C = weight of cage and cars.
Where R = weight of rope.
Where d = small diameter of cone.
Where D = greatest diameter of cone.

Applying this to our proposition we obtain

$$\frac{14 \times [6 + (2 \times 3.5) + (2 \times 4.5)]}{6 + (2 \times 3.5)} = 23.7 \text{ feet}$$

as the large diameter.

Proving this by the same method we have the following result:

A load of $3\frac{1}{2}$ tons on a 23.7 diameter is equal to 5.92 on a 14-foot diameter, and

$$14 - 5.92 = 8.08$$

as the unbalanced load at the lift.

When the back balance reaches the bottom it is 8 tons on a 14-foot diameter, and 8 on 14 is equal to 4.72 on a 23.7-foot diameter, because

$$\frac{8 \times 14}{23.7} = 4.72,$$

and the load at the top of

$$9.5 - 4.72 = 4.78$$

as the unbalanced load at top.

Then an unbalanced load of 8.08 on a 14-foot drum is equal to an unbalanced load of 4.77 on a 23.7-foot drum, because

$$\frac{8.08 \times 14}{23.7} = 4.77,$$

which is the same as found above within $\frac{1}{100}$, due to the fractions not being carried out to five places.

The above proves the formula to be correct, and the required diameter to answer the question is 23.7, or 23 feet and $8\frac{3}{5}$ inches.

The small end of the cone may be considered a lever which is

acted upon by both the load and the back balance alternately when at the bottom, and the large end is a lever acted upon by both the load and the balance when at the top; therefore the diameters should be inversely proportional to the sum of the loads which act on each respectively. The sum of the load and balance which act on the small diameter is equal to

$$W + C + R, \text{ and } C + R.$$

The sum of the weights which act on the large end of the cone is equal to

$$W + C \text{ and } C.$$

The largest weight into the shortest lever is equal to the smallest weight into the longest lever; therefore,

$$[W + C + R + C + R] \times 14 = [W + C + C] \times 23.7,$$

which is the same as the formula given—

$$\frac{d \times (W + 2C + 2R)}{W + 2C} = D.$$

When the large end is given the small end is found by

$$\frac{D \times (W + 2C)}{W + 2C + 2R} = d.$$

2. With an indicated pressure at the boilers of 60 pounds per square inch, what should be the diameter of the steam cylinders?

Ans.—The load to be lifted from the bottom is 8.08 tons = 16,160 pounds. If one engine is on the dead centre, the other must be able to start the load. The circumference of the drum at the lift is

$$14 \times 3.1416 = 43.9824$$

feet; therefore the rope travels with the load 43.98 feet while the piston travels 10 feet. We have, therefore, the simple calculation—

$$\frac{43.9824 \times 16,160}{10} = 71,075.55$$

pounds, the total force required on the piston to overcome the net load, and this divided by the effective pressure on the piston per square inch will give the area.

Assuming one-third of the boiler pressure to overcome fric-

tion, loss between boiler and cylinder, and to compensate for area of piston-rod, etc., the average pressure may be assumed at 40 pounds per square inch; therefore the area of the piston should be

$$\frac{71{,}075}{40} = 1776$$

square inches; this would be a cylinder whose diameter is

$$\sqrt{\frac{1776}{0.7854}} = 47.5$$

inches.

3. Allowing time for changing cars, at top and bottom, and in getting up and slacking off speed, with an average piston speed of 400 feet per minute, how much coal would these engines hoist in a shift of 10 hours?

Ans.—The circumference of the small end of the drum is

$$14 \times 3.1416 = 43.9824,$$

and the circumference of the large end is

$$23.7 \times 3.1416 = 74.4559;$$

therefore the average circumference and lap of rope will be

$$\frac{43.9824 + 74.4559}{2} = 59.219$$

feet each revolution.

$$\frac{1600}{59.22} = 27$$

revolutions for each hoisting, and

$$27 \times 10 = 270 \text{ feet}$$

which the piston has to travel for each run. A speed of 400 feet per minute is

$$\frac{400}{60} = 6.6$$

feet per second, and

$$\frac{270}{6.6} = 40.9$$

seconds is the time required for each hoist.

If we allow 49 seconds for changing the cars and in getting

up and slacking off speed, we have a total time required for each hoist of

$$41 + 49 = 90 \text{ seconds} = 1\tfrac{1}{2} \text{ minutes},$$

which would be equal to 40 hoists per hour, or 400 in a day of 10 hours. The load is 6 tons, therefore the full hoisting capacity is

$$6 \times 400 = 2400 \text{ tons}$$

in a shift of 10 hours.

4.—What is the horse-power of such engines?

Ans.—The cylinders are 47.5 inches = 1776 square inches area. The average piston speed is 400 feet per minute, and the average effective pressure is 40 pounds per square inch.

$$\frac{1776 \times 400 \times 40}{33{,}000} = 861 \text{ H. P.}$$

in each cylinder, and

$$861 \times 2 = 1722 \text{ H. P.}$$

while at that speed and pressure. Practically both speed and pressure could be increased until such engines developed 3000 horse-power.

Ques. 18.—How would you construct the brake on hoisting engines at collieries?

Ans.—The proper type of "brake" would depend on the power and speed of the engines. For powerful engines at high speed I would prefer a steam brake with iron band and lined with wearing segments all round the brake ring; and for engines of moderate power and speed I would prefer block brakes acting on a segment of the brake ring and operated by a hand lever.

The bands are in halves, one over and one under the drum, attached on the opposite side of the drum from the engines by adjusting screws to regulate their length. The other ends are jointed on opposite sides of the centre of motion of the lever; therefore both are tightened or slackened by the one motion of the lever. The band by encircling the drum spreads the pressure and reduces its intensity per unit of surface, lessening the compressive and torsional strain on any given point of the drum; it also gives full contact and conducting surface to carry off the

heat generated by friction; but the motion of the band in tightening or slacking is at right angles to the radius, therefore it requires considerable motion of the lever to produce any effect by the band on the ring. This is an advantage where steam is used to operate the bands, because steam acts suddenly and amplitude of motion imparts elasticity, relieving the brake of the sudden jerk caused by the steam piston. With hand levers this is a disadvantage. If the engineer has to move the lever through an arc 7 or 8 feet to tighten the brake, it is too slow to be efficient.

With a block brake the whole pressure is concentrated on a segment of the circle, requiring strong brake rings to prevent distortion and fracture. The limited conducting surface may be apt to cause heating, but a short movement of the hand lever is sufficient to tighten the brake. The block moves nearly in the direction of the radius, therefore a movement of one-eighth of an inch will apply or release the brake; therefore a movement by the hand of the lever of 3 feet or 36 inches will multiply the leverage 288 times. With such a brake if the engineer apply a force of 50 pounds on the end of the hand lever through an arc of 3 feet, he will apply a pressure of 7.2 tons on the block, because

$$36 \times 8 = 288,$$

and

$$\frac{14{,}400}{288} = 50$$

pounds. As friction is independent of the extent of surface in contact, when the total pressure is the same, the block brake is equally efficient as a brake. Block brakes should be under the drum, so that the pressure upwards will reduce instead of increase the weight upon the drum shaft and consequent friction of the journals in the pedestals.

Ques. 19.—Explain what form of "safety catch" you prefer, and why, as compared with others?

Ans.—Would prefer the catch now generally adopted in the anthracite coal regions of Pennsylvania.

These catches consist of eccentric sectors or cams, having their faces serrated or toothed like a saw, keyed on shafts, which are

placed across the top of the cage frame, one on each side of the guides, a spiral spring around the shafts throws the cam inward against the guides, the springs are held back by connecting rods or chains attached to the main suspending clevis, which connects the rope and cage; therefore the weight of the cage is the force which holds back and prevents the springs from acting. If the rope or chains should break or even slack from the cage resting, the cams are thrown in, the teeth bite into the timber, and the weight of the cage, acting through the cam shafts as a fulcrum, wedges them tight by forcing the increasing throw of the cams between its centre of motion and the grip. The guides being caught on both sides cannot split or be forced out of position. Another form of catch consists of wedge blocks inserted between the guide and the sides of the guide shoe, with the narrow end upwards, having springs or levers to force them up tight when released by the breaking or slacking of the rope. The wedges forced upwards by the springs and the downward motion of the falling cage, wedge the blocks between the shoe and the guide and hold the grip. But if the guides are coated with ice, the wedges have not sufficient friction to catch or break the crust, and allow the cage to jam itself tight on the guide; therefore we would not be in favor of using them, especially in downcast shafts in winter.

Pointed bars working against ratchets and sharp-faced cams have been tried with the view of catching quick before the descending cages acquired momentum to tear or break the guides, but they are found to split and destroy the guides or are broken by the sudden jerk, and are consequently obsolete at well-appointed collieries.

Ques. 20.—Give your views as to whether the so-called safety catches are or are not a safety appliance.

Ans.—In England, where all the coal is raised through shafts, statistics prove that during 11 years only 1 life was lost by the breaking of ropes for every five and one-half million tons of coal raised. Data from the same source show by analysis that a man may ride down and up a shaft on a cage every day in the year, Sunday included, for 6840 years before meeting with a fatal accident from this cause. And although riding in shafts must be

acknowledged as dangerous, still we think that any form of safety catch which can reduce this ratio of danger must be very near perfection.

Where the cage speed is moderate and its weight heavy in proportion to the load, then the danger from descending on the empty cage can hardly be taken into account; but with heavy loads, light cages and high speed, the case is very different.

In deep shafts, with an average speed of 27 to 30 miles an hour in the middle of the run, the cage speed may be 50 and in some cases 60 miles an hour.

With a lightly constructed steel cage, the weight of the empty cars may be more than half the descending load, and when the cage has only a few men on, it may be only about two-thirds of the common descending weight; therefore if the springs of the catches are made just strong enough for the empty cage and car to keep them back when men are descending, the weight may be so much less that the least ice or roughness on the guide, together with the resistance of the air at high speed, may be sufficient partly to slack the rope and cause the cage to stick long enough for the descending rope to drive it down and snap the rope by the jerk from the slack given out. Of course, the springs can be *pinned* back when men are descending, as has been practised at some collieries; but there still remains the danger of this being neglected. If the springs are weak enough to be safe for men descending on the cage at a high speed, they will not have force enough to bite into the guides and be efficient in case of accident.

Of course, we assume that the rope is much more likely to break, with the ascending load; and when the rope breaks the momentum will maintain the cage an appreciable amount of time before it changes direction, and begins to fall back, thus giving the springs time to act and fasten the cage. If the cage chains broke, the catches might hold; but if the rope broke several hundred feet above the cage and fell on top of it, rope, cage guides and all would go to the bottom in a heap, and the rope is more likely to break at the sheave than at the cage, because its own weight is added to the strain. Cage chains can easily be made stronger than the rope, and are convenient for

inspection; therefore, with proper care, their liability to break is very remote.

All things considered, we incline to the opinion that the safety-catch, under ordinary conditions, creates a false sense of security; and we would rather prefer to depend on close inspection and prompt removal of doubtful ropes than on safety-catches. It requires attention to keep them in proper working condition, and they complicate the plant more than they increase safety, and although they have become popular enough to be inserted in the mine law, we have not been able to discover any very good reason why. Experience in the future with high-speed hoisting will probably prove that catches are a source of danger instead of safety.

Ques. 21.—Would you be in favor of using "detaching hooks" on the hoisting plant of a colliery?

Ans.—Accidents from over-hoisting or passing the mark are very rare; therefore they are intended to remove a very remote danger. If the engine was moving slowly, the rope might detach and the cage hang suspended without further damage; but if the cage was moving slowly, the engineer could stop the engine with the break before an accident would result.

If the cage was at a high speed by the engineer neglecting or mistaking the position of the load by the indicator, the hook striking the ring at such a velocity would be more likely to break the detaching ring than to free the rope; if it did detach the rope, the momentum of the cage would still dash it into the head frame, and the result would be a wreck at all events. Detaching hooks must be often cleaned and oiled, the jarring caused by the repeated slacking and tightening of the rope at the lift tends to make the shackle-pin wear a dent in the face of the jaws, which must be kept planed off smooth to be efficient; therefore the detaching hook, like the safety-catch, is seldom in perfect working order.

The strongest arguments in favor of their use have been advanced by those interested in their manufacture. They complicate the rope attachment and add so little, if anything, to security of life or property, that we would not be in favor of using them.

HAULAGE.

Ques. 22.—What grade would you consider proper for the water-level gangways of a colliery?

Ans.—It is evident that a gangway where cars are to be hauled in both directions by animal power should be graded so that the loaded cars could be hauled out with the same power that hauled them in empty. With the average mine car this would be a rise of 1 in about 150, or nearly 2 inches on an 8-yard rail. Engineers generally require the gangways nearer the level than this, but except there be some special reason why the grade should be kept down to the level, on general principles we could never see what was to be gained by "dead levels." The great object is to have an even grade, so that there will be no bumping and uncoupling on the trip, or dead levels where a bad-running car may overpower the team, causing trouble and delay. With very level roads the drain ditch must be kept very clean, or the water will overflow the track and corrode the iron, the road will be muddy, every one connected with the hauling will walk in water and mud, and the expense of moving the coal will be increased proportionately. This is very important in collieries with long roads. If we take some of the mine tracks and compute the tons per mile per day that are transported over them and compare it with the tonnage on the main stems of the different railroads, we will at once see that the mine tracks are entitled to more attention than they generally receive, when the traffic on them is considered.

The track is generally laid by the miners who drive the gangway, and laying the track is included in their contract. They are generally supplied with a plumb-level, having a base of sufficient length to cross the track and show the difference of level between the two rails. It is also used to grade the road. The base of the level is about 4 feet, which is $\frac{1}{6}$ of the length of an 8-yard rail. If the grade is $1\frac{1}{2}$ inches to a rail, the miner grades the track to the level with a chip $\frac{1}{2}$ inch thick under the outer end, which is correct if properly done. When the rail is laid down it is spiked to the ties and the level applied near the inside end; it is then raised up to correspond to the grade, and fine coal tamped under the ties; but the grade is only on the inside

half of the rail. When the end of the rail is raised the weight of the ties bends it down in the middle, leaving the rail dead level at the centre, and graded only at the end where the plumb-level is put on. The reason of this is that miners always try to have it low enough when laid down, as it is much easier to raise than to lower it. There may be exceptions, but this is the general method of laying tracks in gangways, and men on contract cannot afford time to make a perfect job, which accounts for so many poorly laid tracks. To grade a gangway track properly requires two or more lamps placed near the rail, and then sights taken along it for 30 or 40 yards, so as to see whether the grade is uniform, and then raise and tamp it accordingly, as is practised on our railroads. In answer to the question, we would consider 2 inches rise on 8 yards as a proper grade for the average gangway in the anthracite region.

Ques. 23.—Is haulage by a steam locomotive cheaper than by a mule?

Ans.—That depends upon the condition and nature of the work to be performed. The cost of a mule and his driver, including interest and depreciation is about $2.75 per day. He can exert a pull of from 100 to 150 pounds when traveling at a speed of from 2 to 1½ miles per hour and on grades of 1 or 2 per cent. A mule can exert a pull of 300 pounds under the above conditions temporarily or at a slower speed. If the grade is 2 per cent. in favor of the loaded cars, and the friction 1½ per cent. which is about right for a good T-rail track, then the loads would run down without pull. If the empty car weighed 1 ton or 2000 pounds then it would take (2 per cent. + 1½ per cent.) = 3½ per cent. of 2000 pounds = 70 pounds to pull up one empty car. Then one mule could pull 150 ÷ 70 = 2 empty cars up grade. Under ordinary mine conditions, making allowance for delays, we think 12 miles per day is a good average travel for a mule. Of this distance 6 miles is traveled hauling empty cars in, and 6 miles hauling loaded cars out. If each car held 2½ tons then his net ton mileage would be $2\frac{1}{2} \times 2 \times 6 = 30$ ton-miles per day. This figure is within the limits of practice. The cost of hauling one ton of coal one mile then is $\frac{2.75}{30} = 9$ cts. The cost per ton-mile would probably be from

4 to 6 cents if the mules were worked in teams of 3 or more as they are usually worked in the anthracite region.

A ten-ton steam locomotive* (Fig. 102) costs say $5.00 per day including engineer, coupler, fuel, interest and depreciation. The tractive effort is about $\frac{1}{5}$th of the weigth on the drivers at a speed of say 6 miles per hour. As all the weight is on the drivers the tractive effort is $20{,}000 \div 5 = 4000$ lbs. If we were ascending a 2 per cent. grade it would require $(2 + 1\frac{1}{2}) = 3\frac{1}{2}$ per cent. of 20,000 pounds = 700 pounds to move the locomotive itself. This would leave $(4000 - 700) = 3300$ pounds

FIG. 102.

as the force available for pulling empty cars. As it takes 70 lbs. to pull one empty car the locomotive could pull $\frac{3300}{70} = 47$ cars, or it could pull as much as 24 mules. But as locomotives are seldom worked to capacity and the usual condition of grade and tracks is not like those we have assumed, let us suppose that a trip consists of 20 cars. This engine could travel in the ordinary course of its work say 30 miles per day, or 15 miles with a loaded trip. The loaded trip holds $(20 \times 2\frac{1}{2}) = 50$ tons of coal, and the net ton mileage is $(50 \times 15) = 750$ ton-miles per day. The cost per ton-mile is $\frac{5.00}{750} = \frac{67}{100}$ of a cent; which is within the limits of practice. It will therefore be seen that under favorable conditions the cost of haulage with a locomotive would be less than $\frac{1}{10}$th of the cost of mule haulage.

But a locomotive costs more per day than a mule and it re-

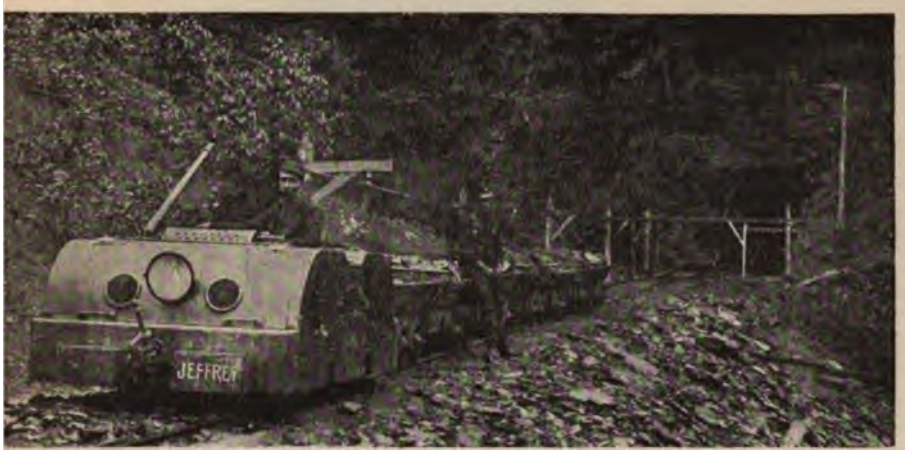
*H. K. Porter Co., Pittsburg, Pa.

quires a better track; so that the cost of locomotive haulage rapidly increases as it loses time in switching, waiting, or running with small trips or on short runs. As there are some kinds of switching at which a locomotive could do little more work than a mule, I think it is doubtful if the mules will ever be entirely displaced in our mines.

Besides this, a steam locomotive can only be used when the ventilation is good and its smoke is not drawn into the working places. But even under favorable conditions of ventilation it is objectionable on account of the smoke and the danger of the fire.

Ques. 24.—What other kinds of locomotives can you describe?

FIG. 103.



Ans.—Electric locomotives (Fig. 193) are extensively used in our mines especially in the bituminous regions. They cost more than steam to install and to operate on account of the generating plant and trolley lines. The daily cost including labor, fuel, supplies, interest and depreciation on locomotive and plant is from $8.00 to $12.00 per day. These are greatly preferable to a steam locomotive under any conditions of ventilation. Although there is no fire on an electric locomotive it is believed that they are capable of firing an explosive mixture of

* The Jeffrey Manufacturing Co., Columbus, Ohio.

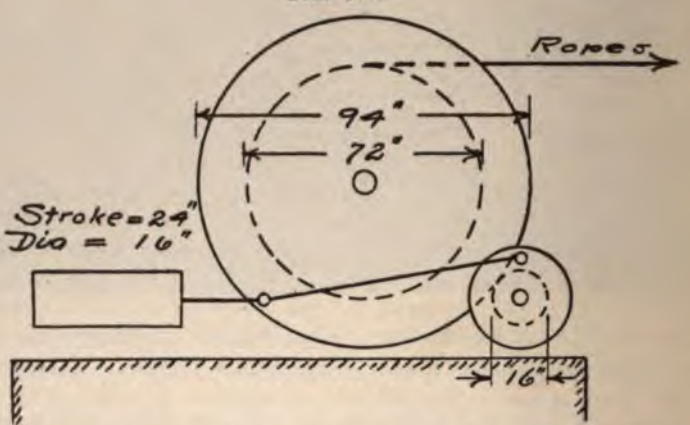
fire damp, and are sometimes objected to on this account. The electricity itself being carried in a naked trolley wire is somewhat of a danger, especially if over 250 volts are carried on the line; and at this voltage it is hardly practicable to carry the elec-

FIG. 104.

tricity over 1½ miles without making a very expensive line or being content with an inefficient operation of the locomotive.

Compressed-air locomotives* (Fig. 104) do away with the pos-

FIG. 105.



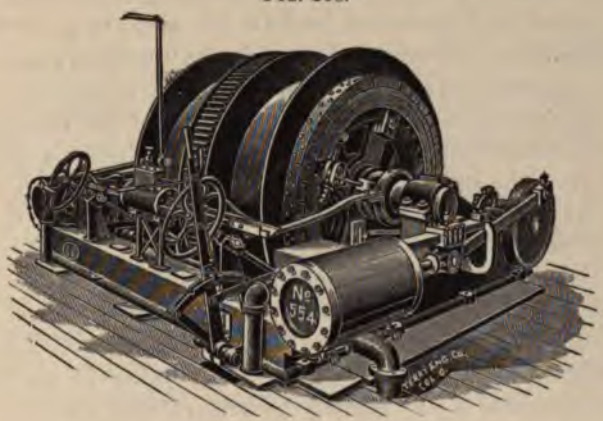
sibility of fireing fire-damp and are successfully in operation in a number of our collieries. These locomotives can run say from 3000 to 6000 feet between charging stations under average conditions; but this depends greatly upon the design, load and so forth. The original cost is somewhat greater than an electric

*H. K. Porter Co., Pittsburg, Pa.

plant for the same work, and the operating expenses are about the same. They require a special air line and compressor system, as the air required by them is delivered at a pressure of about 700 pounds per square inch, and is, therefore, not available for the ordinary purposes for which compressed air is used.

The hauling capacity of both electric and compressed-air locomotives is the same as that of a steam locomotive of equal weight.

FIG. 106.



Ques. 25.—We have a tail-rope hauling engine on hand whose dimensions are shown in Fig. 105, and want to know if it will operate a tail-rope plant to do the following work:

Maximum grade against loads (on the slope)	5 per cent.
Greatest grade against empties	3 per cent.
Average speed in miles per hour	6
Number of cars per trip	25
Weight of empty car	2000 lbs.
Weight of loaded car (gross)	7000 lbs.
Length of haul	3000 ft.
Boiler pressure per sq. in.	70 lbs.
Elevation above tide of top of drum	1250 ft.
Elevation above tide of inside bull wheel	1225 ft.

Figure 106 is a perspective view of our engine.*

* Nelsonville Foundry & Machine Co., Nelsonville, Ohio.

Ans.—First we must find the size of the ropes required. The head rope has to pull 25 loaded cars up a grade of 5 per cent. Now 5 per cent. is the tangent of the angle of the grade but because, for angles up to 10° (= 17.6 per cent. grade) the tangent and sine of the angle are nearly equal, they may be considered equal in rough calculations. Considering 5 per cent. then as the sine of the angle of the grade and the coefficient of friction $1\frac{1}{2}$ per cent., we have the pull of the loaded trip equal to $25 \times 7000 \times (5 + 1\frac{1}{2}$ per cent.) = 11,375 lbs. Assuming the friction of the ropes, rollers and sheaves to be 3000 lbs. we have a total pull of 14,375 lbs. or 7.19 tons. By looking in Roebling's catalogue of wire ropes we find a rope 1″ diameter of extra-strong crucible steel, and having 6 strands of seven wires to the strand, will bear a working load of 7.40 tons, weigh 1.58 pounds per foot and go around a drum or sheave $5\frac{3}{4}$ feet in diameter without injury. As the rope is strong enough and our drum is 6 ft. in diameter we will adopt it for our head rope.

The tail-rope has to draw the empty trip up a grade of 3 per cent., and overcome the friction of the ropes, rollers and sheaves. Then $2000 \times 25 \times (3 + 1\frac{1}{2}$ per cent.) $+ 3000 = 5250$ lbs. = 2.62 tons is the pull on the rope. By looking at Roebling's table we find that a $\frac{5}{8}$th″ rope of the same class as our head rope will work under a strain of 3.02 tons, go around a sheave $3\frac{1}{2}$ feet diameter and weigh .62 lbs. per ft. We will therefore select it for our tail rope.

The friction of the ropes, rollers and sheaves is about $\frac{1}{10}$th of their weights. The time when the greatest weight of rope is "out" is when the loaded trip is starting out. To be on the safe side we will calculate our friction for that weight. The length of the head rope is a little longer than the haul, say 3500 ft. and its weight is $3500 \times 1.58 = 5530$ lbs. The length of the tail-rope is about 6500 ft., but at the time the loaded trip is starting out the length on the road will be only 3500 feet. Its weight will be $3500 \times .62 = 2170$ lbs. If the rollers are 20 feet apart there will be $\frac{3000}{20} = 150$ rollers in the middle of the track and 150 rollers along the rib to carry the tail-rope or 300 rollers in all. As each roller will weigh

about 15 lbs., the weight will be 300 × 15 = 4500 lbs. To this we can add say 1500 lbs. for the bull wheel, and say 10 sheaves @ 100 lbs., each = 1000 lbs. The sum of the moving weights then is:

Head rope	5530 lbs.
Tail-rope	2170 lbs.
Rollers	4500 lbs.
Bull wheel	1500 lbs.
Sheaves	1000 lbs.
Total	14700 lbs.

The friction of the ropes, rollers and sheaves is 14700 ÷ 10 = 1470 lbs., or only about $\frac{1}{2}$ of what we assumed it to be when we called it 3000 pounds.

As the bull wheel is (1250 — 1225) = 25 feet lower than the top of the drum on the engine, there will be a force of 25 × 1.58 = 39 lbs. to overcome the gravity of the head rope when starting out. But the tail-rope will help pull the head rope out by a force equal to 25 ft. × .62 lbs. = 15 lbs., so that there will be a pull of only (39 — 15) = 24 pounds necessary to overcome the resulting gravity of the ropes. The gravity of the ropes is too little in this case to need any allowance, but as the data to find it is given we will include it. When the loaded trip reaches the mouth of the mine the friction is less because most of the head rope is replaced by the lighter tail-rope, and the resultant gravity of the ropes becomes nil, because the tail-rope balances itself. To be on the safe side, however, we will count the friction and gravity as though they were always the same.

The pull at the engine is:

Pull of loaded trip	11375 lbs.
Friction—Ropes, Rollers and Sheaves	1470 lbs.
Gravity of ropes	24 lbs
Total	12869 lbs.

As we figured our rope for 14375 pounds we are on the safe side. Now this engine has two cylinders, and the cranks are set at

right angles, so that when one crank is at a dead centre, the other cylinder, which is at the position of maximum effort, has to bear all the pull. Dropping 10 lbs. off of the boiler pressure for the average pressure of steam in the cylinder we get 60 lbs. The area of the piston is $16^2 \times .7854 = 201$ square inches. The pressure on it is $201 \times 60 = 12060$ lbs. This force exerted at the end of the crank arm, which is 12″, is equal to a force of $\frac{12 \times 12060}{8} = 18100$ lbs. exerted at the teeth of the pinion. The force of 18100 pounds is exerted on the circumference of the big spur wheel, which is 94″ in diameter, and the rope acts on a drum which is 72″ in diameter. But the drum is increased say 12″ in diameter by having the rope around it, so that we call it 84″ diameter. Now the forces exerted at the two ends of a lever are inversely proportioned to the lever arms or:

$$84 : 94 :: 18100 : x.$$

$x = 20{,}300$ lbs. which is the pull the engine is capable of exerting on the rope without making any deduction for its friction. Deducting $\frac{1}{5}$th for the friction of the engine we get 16,240. As this is about 3000 pounds more than the pull required we conclude that the engine can easily do the work as far as pull is concerned.

Six miles per hour $= \frac{6 \times 5280}{60} = 528$ ft. per minute $= 12 \times 528 = 6336$ inches per minute. The circumference of the rope drum is $3.1416 \times 84 = 261$ inches. The speed of the drum is $\frac{6336}{264} = 24$ revolutions per minute. The number of revolutions of the pinion and crank is inversely as the diameter of the pinion and spur wheel, or

$$16 : 94 :: 24 : x.$$

$x = 141$ revolutions per minute.

The average piston speed is $141 \times 4 = 564$ feet per minute which is quite common for this class of machinery. That speed could probably not be reached while the loaded trip was on the slope but a greater speed could be reached with the empties or

when the loads were farther inside. Altogether we think this would be a very suitable engine for the work.

The horse-power exerted while the loaded trip was on the slope would be:

$$\frac{12{,}869 \times 528}{33{,}000} = 206 \text{ H. P.},$$

which would probably require additional boiler plant.

Tail-rope haulage is an excellent system for rough, undulating roads where the grades are too steep for a locomotive to work economically and where the gravity of the empty trip will not carry the rope in. It is extensively used in many parts of the United States, and its simplicity is a great point in its favor. It can be worked so as to go around curves, on several branch roads, and can drop parts of the trip at different points. But all these things are considerably more difficult to perform with a tail-rope system than with a locomotive. As the length of the haul and output increase, the sizes of the trip and rope increase until they are inconvenient to handle.

CHAPTER X.

OPENING COLLIERIES.

Ques. 1.—Show by a diagram how you would open a colliery by a slope on a seam 6 to 10 feet thick, dipping 10° to 20°, explaining the arrangement of the ventilation.

Ans.—Plan No. 1 (Fig. 107) gives a general idea of opening a colliery by a single-track slope, working two lifts, and continuing the slope to open a third. *SS* represents the slope, which is also the intake airway; *AA* the return airways, one on each side of the slope.

The fan *F* is on the top of a shallow shaft sunk to the coal, the airways curving into it; the airway from the other side is bridged over the slope at *K* by slanting up through the strata into the side of the shaft. The shaded curves at each lift on the slope marked *K* show where the branch road is curved round into the strata over the coal to make a proper landing, and at the same time cross over the return airways, making an *overcast* in solid strata. A small hole is drilled through from the drain ditch on each curve to run the water down the airways and along the ditch, across the slope, by the small slant headings into the pump. *P* is a recess for a steam-pump, the suction pipe passing down the airway to the water, and the steam and column pipes passing up the airway to the surface, through a small heading parallel to the slope, the exhaust being inserted in the suction pipes. *W* is a traveling way as an outlet to the surface, *gg* are the main gangways, *vv* the upper or slant gangways, driven in sections to work each group of chambers. The arrows mark the courses of the currents, their being five separate splits of air represented, one to each gangway, and one to ventilate the continuation of the slope for a third lift. The small volume required to supply the slope sinkers is controlled by the regulator at *R*. When the section of airway *A*, being

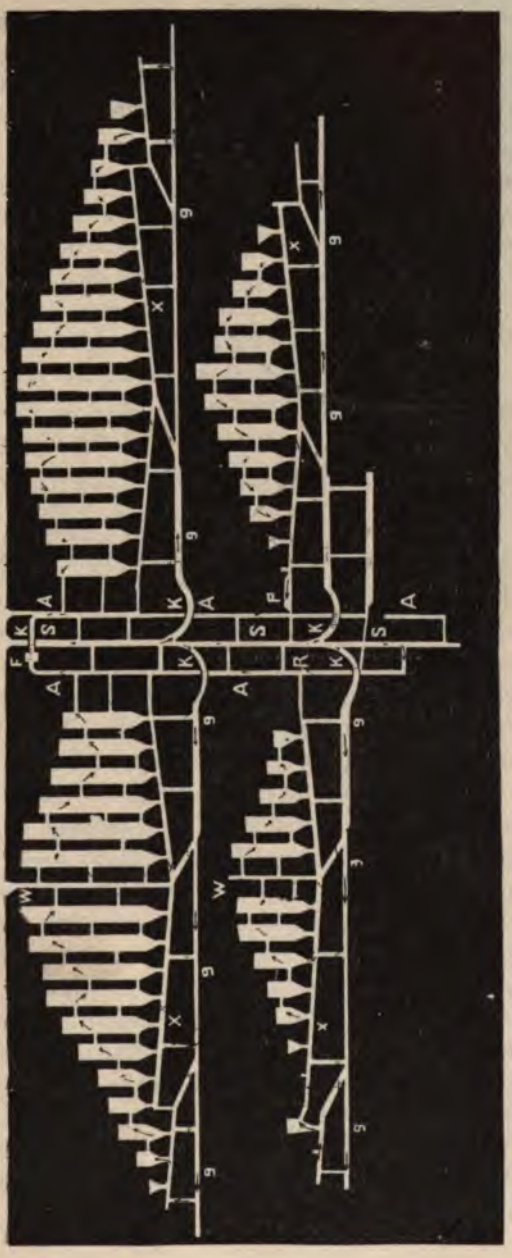


Fig. 107.

Plan No. 1.

driven up connects with the pump heading, the other openings on the slope are closed, and the return from the sinkers passes up the other side by a hole in the stopping at the pump, and so on, changing the current to the cross-heading next the face alternately as the slope face advances down.

The upper gangways *vv* are graded just sufficiently to move the cars, except where they pass through the pillar to the main level; this part is driven on a grade of 6° or 7°; therefore the length of the steep slant will depend on the dip, and would not be convenient on angles above 20°. Assuming 10 yards as the required thickness of pillar between the main and upper gangways on a dip of 20°, the vertical height to be gained by the slant to reach the upper gangway would be

$$30 \text{ feet} \times \text{sine of } 20° = 30 \times 0.34202 = 10.2606 \text{ feet.}$$

To gain this elevation with a grade of 6° would require a length on slant of

$$\frac{\text{Height in feet}}{\text{sine of } 6°} = \frac{10.2606}{0.10453} = 98 \text{ feet} = 32\tfrac{2}{3} \text{ yards;}$$

so that 100 feet of slant on 6° will make a good road for all practical purposes. The gangways from the slope are not opposite each other, but branch off so as to blind each other, thus avoiding a large opening to weaken the top. The single track admits of cars being run to each gangway, and for easy dips is more convenient than double tracks; the quantity to be hoisted is only a question of power and speed, as from 1 to 6 cars can be handled at a time, the loss from want of back balance is more than compensated for by the simplicity and convenience of the whole plant; the slope is also much stronger, cheaper, and easier repaired.

By this plan the main gangways form continuous intake air courses, only broken while a trip is passing in the slant, which can be avoided by double doors; regulators in the door-frames can regulate the volume to be carried to the gangway face, and thus cause partial splitting into each slant, to clear the smoke. No inside turnouts are required, as the trip of cars in one slant can be run down after the empty trip passes into the next inside slant, and so on alternately.

As the chambers reach up parallel with the airways along the slope, they are connected by headings through the pillar, causing the fan to exhaust from the upper end and highest point of each lift, thus making the ventilation practically *ascensional*.

The same general plan can be applied in shafts by self-acting planes from the different lifts to the shaft level; but with workings to the *rise* of a shaft, the intake current is up and the return current down hill again to reach the shaft level; therefore the ventilation of a shaft cannot be ascensional like slope workings.

Ques. 2.—With diagrams describe the method of opening "Breasts" on steep angles, explaining the different plans, with the conditions to which they are best adapted, comparing the relative cost of each.

Ans.—In mining parlance the term "steep" is supposed to mean dips of more than 30°, because 30° is about the limit on which the chambers can be worked empty. From 30° down to about 25° coal will move by its own weight, but not so violently as to require a battery or obstruction to control and prevent it rushing on to the gangway; therefore the coal is loaded as it is mined, except what may stick alongside the pillars, to afford footing to the miner in reaching the face. From 25° down to 15° the coal will not move except on sheet iron, and as our question refers only to *steep angles* it is only with breasts worked full that we have to deal in this proposition.

Where the angle of dip is over 30° the coal when loose rushes down with great force; the miner must have loose coal up to the face upon which to stand while at work. Where the coal is soft and the seam thin so that the top can be propped, mining has sometimes been done on batteries or dams of plank near the face supported by the props, leaving the whole breast empty below; but in hard seams shots throw the coal with such force that the props, battery and all are often swept away. This mode of working is now obsolete, the danger of working and damage to the coal by smashing down the empty breast having led to its abandonment. The breasts are, therefore, kept full of coal on which the miner works at the face. A manway constructed of props stood slant against the side and clad with plank is built

up on each side as a travelling way to the face and for an intake and return airway, the whole width between being kept full of loose coal.

To keep this mass of coal (which sometimes amounts to hundreds of tons) under control and prevent it from rushing into and choking up the gangway, the breasts are opened from *chutes*. A chute is a narrow opening, from 6 to 10 feet wide, and from 8 to 12 yards long, with a strong *battery* or regulator built of round timber through which the coal can be tapped or run out as required. The plan of chutes and batteries is varied to suit conditions due to steepness of dip, to the character of the top, to the hardness of the coal, and especially to the quantity of gas to be encountered and the volume of ventilation necessary for its removal.

The following diagrams will illustrate some of the plans of opening breasts from the gangway as practised in the *anthracite* coal region of Pennsylvania.

In comparing the cost of the different plans I have necessarily estimated from a fixed basis of price for labor and material, wishing it distinctly understood, however, that I am expressing no opinion as to what should be the price of either labor or material.

For the purpose of contrasting the relative expense of the different plans, we will take the following table of prices as a basis for calculation:

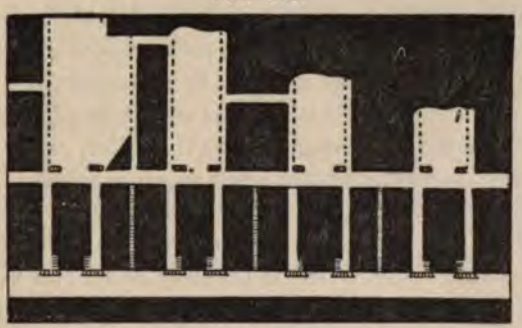
Chute 6′x9′, timber 5 feet from the centres 8″ diameter, plank division on props 3′ from side. Check-stick for loader, with spout, miners to load coal, per yard	$3 00
Chute 6x6, timber 5′ apart, per yard	2 50
Manway up through pillar, 3x3, timbered, per yard	2 00
Heading 5x6 through pillar, timbered, per yard	2 50
Heading across bottom of breasts, 5x5, without timber, but secured to keep open, per yard	2 00
Main battery in breasts, each	4 00
Airway over chutes 5 feet collar, 7 feet spread, 6½ feet clear, timber 8″ diameter and 5 feet from centre to centre. No measurement for cross-holes, per yard	3 50
Round timber 10 inch diameter, per lineal foot	0 15
Round timber 8 inch diameter, per lineal foot	0 10
2-inch-planking for manways, per square foot	0 03

Brattice boards, per square foot 0 01½
Gangsways, 7 feet collar, 11 feet spread, 8½ feet clear of rail, timber 10″ diameter, 5′ from centres, track ties, 3 feet from centres, drain ditch close to lower side leg, and 9″ below bottom of track, per yard 7 00

The above are all supposed to be double-timbered except across bottoms of breasts, which are temporarily secured to keep open. Where no timber is required the coal is so much harder and expensive for blasting powder that the same price is assumed for labor; the employer is supposed to prepare the timber.

Plan No. 2 (Fig. 108) is sometimes adopted on thick seams where the dip ranges from 40° to 60°, with breasts and pillars

Fig. 108.



Plan No. 2.

of 30 feet in width each. As shown in the diagram, there is a chute on each side of the breast. A manway is driven up from the gangway to the heading as a travelling way, no larger than necessary to get up timber, and closed by a trap-door to maintain ventilation. After the first pillar heading is open the chute heading can be used to store timber and obviate the necessity of piling it on the gangway to the danger of drivers and loaders. When the pillars are to be skipped, this manway can be converted into a chute for either side of the pillar; all things considered, it is worth the extra expense. If the travelling way is made in the chute, it requires a plank division, must be one-half wider, requires heavy timber, and in strong coal will cost

nearly as much in the kind of seam to which this plan is adapted.

The advantages claimed for this plan are: Where the coal is hard the chutes require no timber; being only 6 feet wide, the absence of timber allows the coal to pack closer and renders the chute more air-tight when full. Across the mouth of the chute is placed a strong piece of timber as a "booming-stick" to prevent large pieces from getting into the chute, and holds them convenient for breaking, to pass the loader's break-stick. If one battery should be blocked so as to be dangerous for the starter, by going around to the other he can sometimes start both. The starter can generally start two batteries by once climbing up into the heading. When coal is in danger of being covered by a fall, two cars can be loaded from the same breast at once, or one side of the breast can be loaded and drawn much lower than the other, so that when rock falls it will slide into the empty side, leaving the coal free to be loaded from the other side; this way of managing is of great service in pillar skipping. The chute and battery being close to the pillar, serve for skipping without alteration, except a few laggings across the bottom of the old manway, to run the coal behind the battery and shield the starter. The skip is driven slant into the pillar, above the battery, and a small manway chute run up straight from the travelling way as a chute to the skip manway, as shown on the plan.

The objections to this plan are that it draws the sides of the breast more than the middle, and is more likely to damage or displace the manway by the movement of the coal against it. It makes more openings from the gangway, which weaken the pillars; and takes more labor to close with stoppings for ventilation. The greatest objection, however, is its cost, requiring two chutes, two batteries, and a manway in the pillar for each breast.

In comparing the expense of the different plans, it will be necessary to know the value of the sets of timber required in each size of opening, together with what is required for the battery.

A set of gangway timbers requires about 9 feet for a collar and 9 feet for each leg, or 27 feet at 15 cents per foot = $4.05

per set; airway timber will require about 7 feet for collar and legs, or 21 feet at 10 cents for each set = $2.10. Chutes 10 x 6 will require for collar 10 feet and 7 feet for each leg = 24 feet at 10 cents per foot, or $2.40 per set. Chutes 6 x 6 and main headings will require for a collar about 6 feet, and say 7 feet for each leg, or 20 feet at 10 cents per foot = $2 per set. A good strong battery and booming-stick will require say 3 collars of 8 feet each and 3 legs say 7 feet each, and a stick 9 feet, or 54 feet of 10-inch timber at 15 cents per foot = $8.10.

So that we call the gangway timber	$4 00
The airway, chute, and heading timber	2 00
And the battery timber	8 00

Add for laggings on the gangway at 2 cents each, say $1; 50 cents for the chutes and headings, and $1 for slabs to cover the battery, and we have

Gangway timber	$5 00
Chute and heading timber	2 50
Battery timber	9 00

which may include nails, spikes, etc. Taking these values as a basis, to open a breast on plan No. 2 (Fig. 108) will cost for labor and material about as follows:—

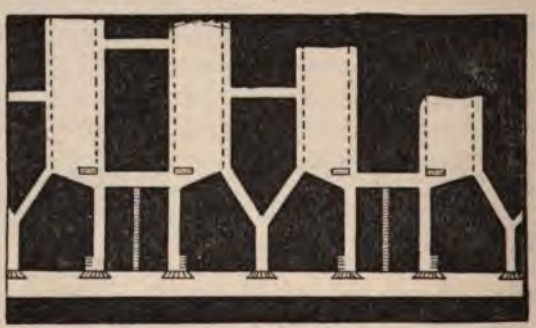
Two chutes 6 x 6, 10 yards each, at $2.50 per yard	$50 00
Two yards, 3 feet extra width for recess below check-stick, 50 cents extra per yard on each	2 00
Manway in gangway stump 3 x 3, secured and stopped, with air door, at $2.00 per yard	20 00
Pillar heading 5 x 6, 10 yards at $2.50	25 00
Heading across to open the breast, 10 yards at $2 per yard	20 00
Two batteries at $4 each	8 00
The chutes will require 5 sets timber, each at $2.50	25 00
Heading requires 6 sets at $2.50	15 00
2 batteries at $9 each for timber	18 00
Loading platform 6 x 6 and sides, each 50 square feet of plank at 3 cents	1 50
Total cost	$184 50

The main chutes are in pairs as shown on the diagram (Fig. 108).

Plan No. 3 (Fig. 109) is a better method for maintaining the air current where gas has to be dealt with, there being only one draw chute to each breast, and therefore but one battery for the air to leak through. Each alternate pillar is pierced by a pair of slant chutes to receive the manway coal, an air check battery being fixed below their junction to keep the current in the breasts.

The manway coal from the other side goes into the main chute under the battery. In the pillar between each pair of main chutes a small traveling way up to the heading serves the same purposes as in plan No. 2 (Fig. 108). This plan may be objected to for drawing the coal only at one side, but that can be remedied by slanting the breast off and bringing the center

FIG. 109.



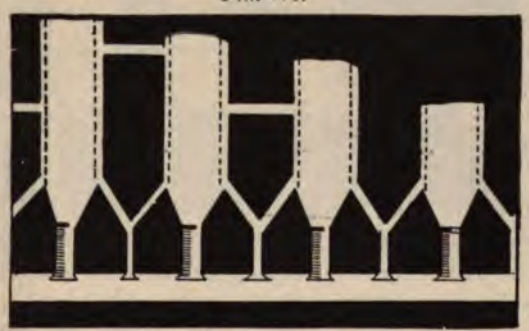
Plan No. 3.

nearer the battery; but if it does move only one side, it leaves the other better as an air manway when stuffed with hay to make it carry the current up. With an airway driven across over the chutes with the cross-holes opened from the junction of the slant manway chutes, by leaving one open to the gangway, and connecting the next to the airway alternately, each pair of breasts will have a separate split of air, and the number of breasts on each split can be made 4, 6, 8, etc., by merely closing the proper cross-holes. This is a good plan of opening breasts on a thick seam where the coal is strong and gas is encountered in dangerous volumes.

On the same basis of prices the expense of opening on this plan will be about as follows:

One main chute 6′ x 6′, 10 yards at $2.50	$25 00
2-yard recess at 50c. per yard	1 00
Examination of the plan will show that only ½ of the pillar heading, ½ of the slant chutes and ½ of the traveling way belongs to each breast, therefore,	
Heading 5 x 6, 5 yards at $2.50	12 50
Manway 3 x 3, 5 yards at $2	10 00
The slant chutes are about 6 yards long and join 5 yards up, therefore each breast has 6 + 2½ = 8½ yards 6 x 5 at $2.50	21 25
One battery	4 00
Large untimbered opening across to pass the air through bottom of breast to meet slant, about 8 yards at $2	16 00
Total for labor	$89 75
Brought forward	$89 75
5 sets chute timber at $2.50	12 50
3 sets heading timber at $2.50	7 50
5 sets slant chute timber at $2.50	12 50
1 battery timber at $9	9 00
Plank for platform and air doors	2 00
Total for opening breast	$133 25

FIG. 110.



Plan No. 4.

Plan No. 4 (Fig. 110) is very generally adopted, and is, perhaps, the best for seams lying on very steep dips, where the coal is *free*, apt to *run*, and gives off large quantities of gas.

The chute is driven up opposite the middle of the breast, 9 or

10 feet wide, and about 8 yards long. The main battery is placed at the top. A manway is divided off on one side with props and plank, about 3 feet wide, as a way to the front of the battery for the starter. From the battery up, the breast is widened out to its full breadth in 5 or 6 yards. The manway chute is driven up in the centre of the pillar about 6 x 6 for 6 yards, and then branched off slant in both directions till it pierces into each breast at their full breadth. From the top of these slants the manways on each side continue up the breasts, as shown on the diagram (Fig. 110).

The advantages claimed for this plan are that only 2 openings are required for each breast, that the loading and starting can be done together, and the starter is not required to cross under the manway to reach the battery. The manway coal is loaded through a slide in the air battery, which is placed below the junction of the slants. The main chute being opposite the centre, draws the middle of the breast and causes less movement of the coal against the manways to damage or displace them. Where the seam is very thick, very steep and very gassy, and the top bench and pillars apt to *run* when moving the coal through the main battery, then the breast can be worked in the top part only, and the coal run down the slant chutes without moving the coal in the breast to endanger the manways. When the breast is worked up the stock coal can be loaded out of both chutes if necessary.

Where the seam is thick enough to admit of an airway being driven across over the chutes, the cross-holes connect at the junction of the slants with an exhaust fan; any leakage of air will be upwards from the gangway and through the main battery, so that if a rush of gas comes down, by moving the battery the current will carry it up through, and away from the starter; it is, therefore, one of the best plans that can be adopted where gas is evolved in large quantities, because, like plan No. 3 (Fig. 109) the breasts can be ventilated in *pairs* or *groups* to suit conditions by merely closing the proper cross-holes, and as the coal at the bottom of the manway is left almost undisturbed by the rush through the battery, it remains practically air-tight, and forces the current up instead of across through the loose coal.

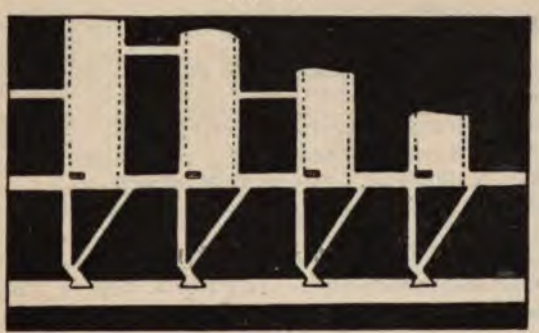
The widening out of the breast above the battery is generally measured full length and paid breast price, to compensate for the disadvantage of working up until it breaks open into the slant chutes, and the air-current connected across the face.

The cost of opening on this plan will be about as follows:

1 chute, 9′x6″, 8 yards at $3 per yard	$24 00
2 slant chutes, 6′x6′; 6 yards each, at $2.50	30 00
1 manway chute, 6′x6′, 6 yards at $2.50	15 00
Widening breast, say	20 00
1 battery at $4	4 00
4 sets of timber for chute at $2.50	10 00
10 sets for slants and chutes at $2.50	25 00
Timber for battery	9 00
Plank for platforms and division in chute, 240 square feet at 3 cents per foot	7 20
Total for breast	$144 20

Plan No. 5 (Fig. 111) is adapted to steep angles where the seam is too thin for an airway to be driven across over the

FIG 111.



Plan No. 5.

chutes, where the breasts are, say 9 and the pillars 7 yards wide respectively.

The chute is driven slant for about 9 feet and then turned straight up the *pitch*, under the side of the breast. The battery is over the mouth of the chute and the coal rests on the gangway pillar, the manway coal on that side runs under the battery

into the chute, the check battery for the loader being placed just outside of the bend at the bottom. The steep part packs full of coal and prevents the air from leaking through the chute, while the slant bottom affords a convenient place to work the check battery. The small slant chute is driven from the front of the check to reach the heading on the other side of the breasts down which the other manway coal is run to the same platform, and is kept air-tight by a stopping, with a slide to run the coal, and a man-door to travel through.

The advantages claimed for this plan are that on very steep dips with a thin seam the ventilation depends on the chutes being kept air-tight, and by this plan the chute being like a shaft packs full if the battery is kept running, and the slant bottom allows the loader full control at the check battery. The drawing at one side allows fine coal to settle against the bottom of the other manway and keeps it from leaking. There is only one opening from the gangway for each breast, and the slant is a better traveling way than if straight with the dip. When properly fixed this plan works well on steep dips.

The cost of opening a breast on this plan, under the above conditions, is as follows:—

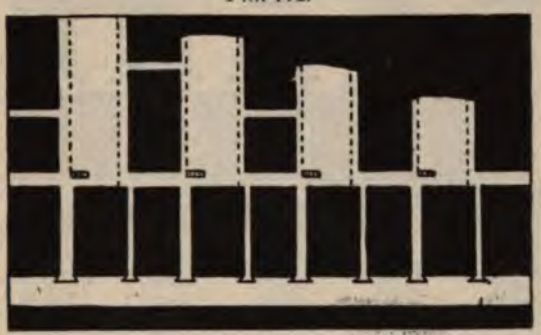
Main chute 6 x 6, from center of platform to battery, 10 yards at $2.50 per yard	$25 00
Two yards bottom, extra width, at 50 cents per yard	1 00
Slant chute 4 x 4, 8 yards at $2 per yard	16 00
Heading through pillar 6 x 6, 7 yards at $2 50 per yard	17 50
Heading across to open breast (no timber) 6 x 6, 9 yards at $2 per yard	18 00
Main battery	4 00
Six sets chute timber at $2.50 per set	15 00
Four sets heading timber at $2.50 per set	10 00
Five small sets in manway slant at $2 per set	10 00
Timber for main battery	9 00
Forty square feet of plank for platform, doors, etc	1 20
Total expenses	$126 70

Plan No. 6 (Fig. 112) is nearly the same as No. 5 (Fig. 111), only modified to suit where the angle of dip is not so steep. The only difference is that the manway chute is driven straight

up from the gangway, because if the slant was on a moderate dip it would not carry the coal without sheet-iron, and that would not suit for traveling on.

The cost of opening on this plan is about the same as plan No. 5 (Fig. 111). If the manway chute is shortened a yard by going square up through the pillar, the wide space in front of the check battery will cost a little more, so that the expense of

FIG. 112.



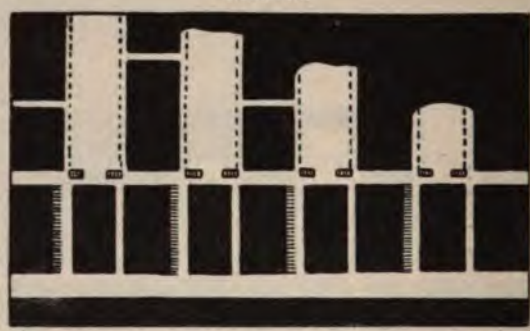
Plan No. 6.

both are about equal; the difference of timber is only the plank for two instead of one platform.

Plan No. 7 (Fig. 113) is best adapted to very thick and strong seams on dips of from 40° to 50°, the soundness of the top allowing the breasts to be worked 12 or 14 yards wide. There is a chute on each side of a breast. One is a close chute 6 x 6, and the other 9 x 6, with a 3-foot traveling way divided off by plank to reach the heading; the close chute having 2 yards at the bottom 3 feet wider to make room at the check for loading. The manway up the side of the chute may be a little cheaper than when driven up between the breasts in the solid, but it is not so air-tight as close chutes full of coal, the planks being more apt to leak, but it makes one less opening and does not weaken the gangway pillar to the same extent. At the top of the traveling way a plank cover is placed slanting from the corner of the pillar to guide the manway coal under the battery and protect the traveling way from coal falling down from the breast manway above.

This method of opening breasts is adapted to about the same conditions as plan No. 2 (Fig. 108), but the way up for the men is neither so safe nor convenient, and is not so tight for keeping the air current in the breasts.

FIG. 113.



Plan No. 7.

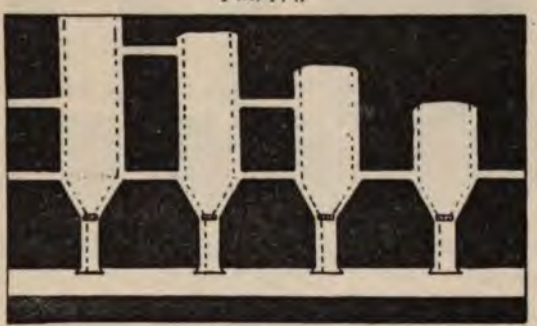
The relative cost of opening by this plan is about as follows:—

One chute 6 x 9, 10 yards at $3 per yard	$30 00
One chute 6 x 6, 10 yards at $2.50 per yard	25 00
Two yards extra width at bottom, at 50 cents per yard	1 00
Heading across pillar, 5 x 6, 10 yards at $2.50 per yard	25 00
Heading across to open breast, 10 yards at $2 per yard	20 00
Two batteries, $4 each	8 00
Ten sets chute timber at $2.50 per set	25 00
Six sets heading timber, at $2.50 per set	15 00
Timber for 2 batteries	18 00
Plank partition, 6 x 30, 180 feet at 3 cents per foot	5 40
Two platforms, 6 x 6 and 6 x 9, 90 square feet at 3 cents per foot	3 70
Total cost	$175 10

Plan No. 8 (Fig. 114) is often adopted on moderate dips. There is only one chute to each breast; the chute is driven up about 7 or 8 yards, 6 x 9, to where the battery is placed. The breast is then widened out on both sides, with the shoulders slanting to nearly full width, where the headings are put through the pillars and the ventilation connected. A plank division 3 feet from the side forms a manway up to the battery, and along

the slant side up to the heading for a traveling way. The manway on the other side starts from above the battery and below the heading, so that the manway coal will pass and not block the mouth of the heading. As the manways start from each side of the main battery it is difficult to prevent leakage. If the dip is over 35°, the weight of the loose coal is on the battery timber; therefore large timber is required to make it secure. The plan is best suited for dips under 35°, or where the coal will just move by its own weight. If the seam is thin and no airway can be driven to conduct the current to the inside places, then the batteries require canvas aprons to cover and keep them tight, and as these are dependent on the loaders for adjustment

FIG. 114.



Plan No. 8.

they are often neglected. On a gangway, say with 10 breasts, if the current is taken in along the gangway and returns through the breasts with a current of 12,000 cubic feet per minute, 1000 feet may leak through each chute and still 2000 be left to reach the gangway and inside chute, which would be better than taking it all to the inside end; but few gangways have one-half that volume of air. With 3000 cubic feet and five breasts, a chute left open may pass the whole current and none reach the gangway or chutes inside. Therefore, this plan is not suited for long tiers of working breasts in a thin seam where gas is plenty and air scant. Its strongest recommendation is cheapness; although it is as good as any other plan on dips requiring sheet-iron.

Assuming that the driving of the chutes includes the opening of the breast to connect the air currents, and the measurement is up to the line of the heading and also across the breast, which is the actual ground to be driven, then the relative cost would be about as follows:—

Chute 6 x 9, 10 yards, at $3 per yard	$30 00
Heading through pillar, 6 x 5, 8 yards, at $2.50 per yard	20 00
Heading across breast (no timber), 6 x 6, 10 yards at $2 per yard	20 00
One battery at $4	4 00
Four sets chute timber at $2.50 per set	10 00
Five sets heading timber at $2.50 per set	12 50
Planking 6 x 24, 144 feet at 3 cents per foot	4 32
Platform, 9 x 6, planking, 54 feet at 3 cents per foot	1 62
Battery timber	9 00
Total cost	$111 44

On moderate dips where the breasts can be worked empty, the miner is then paid by the car, and the opening of the breast is also paid by the car and yard both.

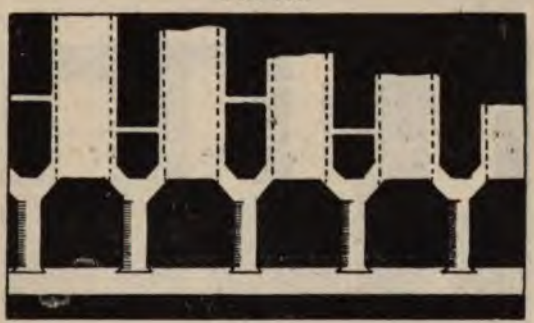
Plan No. 9 (Fig. 115) is seldom adopted, and for general purposes has little to recommend it.

Where the seam is thin, the top weak, the pillars 5 or 6 yards, and the breasts 9; where the top falls and the bottom is apt to slide off and mix with the coal by drawing the breast, this plan may be satisfactory, but only where the ventilation does not depend on the tightness of chutes and especially where the shortness of the lift renders cheapness an important consideration. Under such conditions the plan is to run all the extra coal from the face of the breast down the manways, so that the coal in the breast is not moved until it is finished; then it is loaded as fast as possible at both chutes before the top falls and mixes with the coal. As shown on the diagram (Fig. 115), the chute is opposite the pillar, is driven up 6′ x 9′ for about 8 yards, with a manway 3 feet wide divided off by a plank partition; it is then widened out on both sides into each breast, so that coal from both can be run down the one chute. The top of the chute manway is covered by a man-door to keep out the coal and hold in the air-current; the batteries are only strong "juggler" props

which separate the breast from the manway, with a drawhole left at the bottom; when the coal rushes down to fill the chute it does not fill the slant, it not being steep enough; therefore; the miner can climb from the top of the chute manway to the bottom of the breast manway on the loose coal, or on a plank laid on "juggler" props placed across the chute. The loose coal rests on the pillar, and the starter can fill the chute or draw from both when necessary.

This plan is perhaps best suited for small operations above water-level on thin, steep seams where an airway to the surface

FIG. 115.



Plan No. 9.

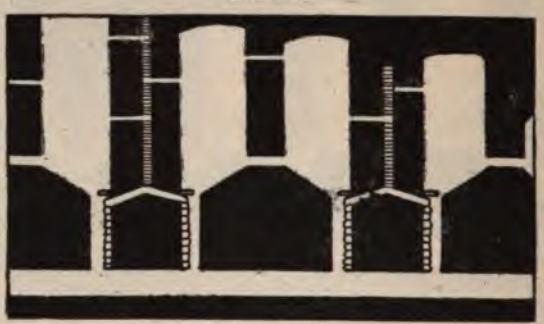
costs little and the breasts are so short that it will not pay to spend much in opening them, this being a kind of temporary arrangement to suit such conditions.

The cost of opening breasts on this plan is as follows: the batteries being only three heavy "juggler props" close together, may be both considered equal to one battery, then

1 chute 6′ x 9′, 8 yards at $3	$24 00
2 slants, each 6 x 6, and 3 yards at $2.50	15 00
Heading to open breast, 6 x 6, no timber, 10 yds. at $2 per yard	20 00
1 battery	4 00
4 sets chute timber and 3 sets heading timber in each slant = 10 sets at $2.50	25 00
Battery timber, $9	9 00
Planking 6 × 24 + 6 × 9 = 198 square feet at 3 cents	5 94
Total cost	$102 94

Plan No. 10 (Fig. 116) dispenses with breast manways by driving a traveling way up through the center of every alternate pillar, thus making one manway serve 2 breasts. This plan of working breasts is sometimes adopted in thick seams where the dip is under 50° and pillars and top strong; it saves the expense of timber for breast manways, but requires more pillar headings, so that the expense of working the breasts is about equal; but the air is not kept so close to the face, and the miner has only the pillar heading as a refuge from danger; therefore, it is not suited for coal which gives off much gas. It will not be satisfactory on steep dips, because the coal in the breast does not *tail*

FIG. 116.



Plan No. 10.

far back, but fills square up and closes the pillar headings, although they are driven next the roof.

We have seen this plan tried where the bottom, to the depth of 18 inches, became loose and slid off among the coal; the sinking of holes on both sides for juggler props helped to cut the bottom loose, and the manway was driven in the pillar to obviate that difficulty, but it effected little improvement, cost more and injured the ventilation, and was therefore finally abandoned. It was found that with 7 to 9 breasts working in one tier, there was always some miner who had his pillar heading blocked, so that it was neither efficient nor convenient for dealing with gas, and on angles above 50° with seams less than 10 feet thick, it is impracticable, and in the opening of the chutes a heading had to be driven from every alternate chute,

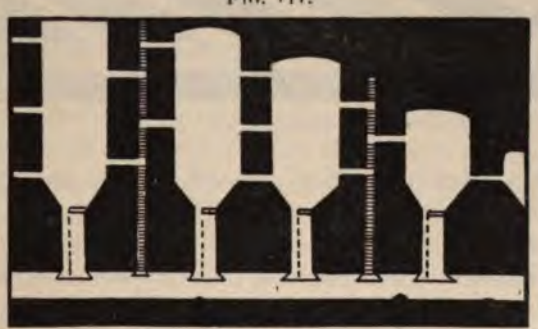
nearly the width of both breasts, and the "blind" pillar between them, before the air connection could be got through and the ventilation established. This was a great inconvenience where much gas was given off, as it required a hand fan to keep it ventilated until the connection was made. Except for special conditions, this plan of working breasts is now obsolete.

The relative cost of opening on this plan will be about as follows:—

Chute 9′ x 6′ with traveling way divided off by plank, 10 yards at $3 per yard	$30 00
Heading across ½ of 10-yard pillar, 5 x 6, 5 yards at $2.50	12 50
Heading across breast from side of 9-foot chute about 7 yards, no timber, at $2	14 00
Heading through ½ of 6-yard blind pillar, 6 x 6, timbered, at $2.50	7 50
Battery	4 00
5 sets chute and 5 sets heading timber to secure the half of each pillar equals 10 sets at $2.50	25 00
Planking 6 × 30 + 6 × 9 = 234 square feet at 3 cents	7 02
Timber for battery	9 00
Total cost	$109 02

Where an airway can be driven over the chutes the breasts

Fig. 117.



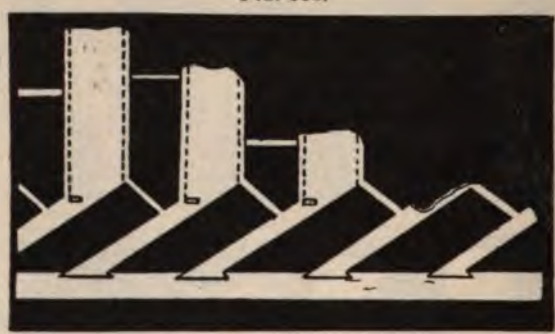
Plan No. 11.

can be ventilated in pairs or groups by connecting the fan to alternate pillar manways so that the air would be exhausted from the upper end of the breasts into the one set of manways and the air find its way up the others.

Plan No. 11 (Fig. 117) is nearly the same as plan No. 10

(Fig. 116), only the manway is driven straight up from the gangway, which involves an additional expense of 10 yards at $2 = $20. The chutes being opposite the middle of the breast, there is a shorter distance to be driven before the connection for the air current can be made. The manway chute is more convenient for the coal mined in it, but being in the top part of the seam while the chute is on the bottom, there is always a step or offset to be made in the manway, while in plan No. 10 (Fig. 116) this is overcome by slanting both halves of the heading up through the seam, which also carries the coal into the chute and prevents it from running out and obstructing the track by the miners traveling in it. This plan is about equal in all respects to plan No. 10 (Fig. 116), but costs $20 more.

FIG. 118.



Plan No. 12.

Plan No. 12 (Fig. 118) is adapted to steep dips, 60° up to perpendicular, with conditions which do not admit of wide breasts or pillars, and the coal free in running through the battery. As shown on the diagram, the chute is driven across the strike of the seam to reduce the grade to about 25°, so that the coal will just move down by its own weight.

From the front of the battery the heading is slanted through the pillar to form the manway chute of the next breast. A plank battery with a mandoor next the roof and a drawhole in the bottom serves as a check and air battery to load the coal from, and at the same time hold the air current in the breasts;

as the coal lies only on the bottom of the chute the miners can travel up and down over the loose coal. Care must be taken to grade the chutes properly, because if they are too steep the coal will fill the chute and block the miners from traveling until it is loaded. With breasts 8 and pillars 6 yards wide respectively, cost of opening breasts on this plan would be about as follows:

Chute 6 x 6, 8 yards at $2 50 a yard	$20 00
Slant heading through pillar, 9 yards at $2.50	22 50
Heading slant across to open bottom of breast, 6 x 6, without timber, 8 yards at $2 per yard	16 00
Battery	4 00
4 sets chute and 4 sets heading timber at $2.50 per set	20 00
Battery timber	9 00
Planking, 6 × 6 + platform 6 × 6 = 72 square feet at 3 cents	2 16
Total expense	$93 66

This plan is cheaper than others, but is not adapted to handling gas or large quantities of coal, and the manway coal has to be loaded regularly, as a carload is about all that they will contain without blocking the air until the second pillar heading is open. This plan is best suited for small operations on a very steep seam, where chutes square up the pitch would be too steep for convenient loading.

One good feature in this plan is that if emptying the breast causes the pillar to *run* or falls should choke the bottom, thus blocking up the return air-course, a new heading can be driven level from below the battery across to the next heading for the current to pass, leaving the triangular stump formed under the breast to hold the fall.

Ques. 3.—Where the dip is so steep that the breasts have to be kept full of coal, in what position would you drive an airway?

Ans.—I would drive the airway over the chutes and as high as the thickness of the seam would permit, the cross-holes connecting near the upper end of the manway chute. The airway being the return, the leakage is upwards through the battery and tends to carry gas away from the *starter*. If a battery should run open, the air from the gangway rushes up through, and

clears out any gas which may be in the gob, and prevents it being driven down on the gangway by the next rush. Driving the airway on the dip side of the gangway just high enough to run the coal into the car will not fulfil these conditions, and loading the chutes on both sides of the track is very dangerous for the drivers.

Ques. 4.—Explain with the aid of a diagram how you would open *breasts* on a seam 6 feet thick, dipping 75°, giving off large quantities of gas, the coal being so *free* that emptying the breasts causes the pillars to run and choke up the abandoned workings, the seam being opened from a tunnel.

Ans.—So far as the mere mining is concerned, the breasts might be opened on different plans; the chief difficulty under the given conditions would be in maintaining the ventilation. A seam 6 feet thick would not admit of an airway being driven over the chutes or the gangway made wide enough for a brattice; even if space was blasted out of the rock the brattice would not serve for a long distance, because of leakage; working two short lifts and making the lower the *intake* and the upper the *return* would not be safe because of the chain pillar running out. To use a forcing fan to blow inwards along the gangways and displace the gas upwards through the broken waste to escape by crop falls could not be depended on.

To drive the gangway and airway over it to the boundary, and then work backwards, with the solid airway as a return, would be a safe plan, but would involve large outlay, and long delay for return of the investment.

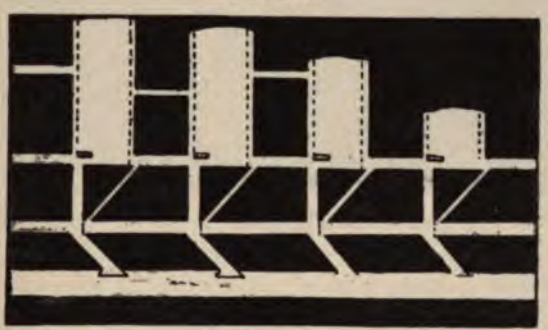
If the gangway was from a slope, the airway could be driven below it by having a second landing below the main bottom from which to hoist the airway coal; but this would require double track the whole distance. Of course, the airway below could be driven with a small hand-car which could be hoisted through slant cross-holes at proper intervals and dumped into the gangway car. With such an airway as a *return*, the gas would have to be forced downwards, but it would answer the purpose if from a slope, but would be impracticable from a tunnel. The airway if below the level of the tunnel, would require a pump, and a pump at the inside end of a tunnel is almost impracticable. The

water from such an airway could be run to the main pump by a *siphon*, but when the water rose in the pump the siphon would cease to flow.

One gangway could be driven above the other by grading up until thickness for a chain pillar was gained. But in a seam 6 feet thick, one would have to be driven in the rock where the upper gangway graded up and crossed over on top of the dividing pillar. Both gangways would require a track the full length, which would be expensive for a thin seam. If the upper track was cut off in sections to save rail, the grading up in the rock would be as much expense as the rail, and could not be sold for scrap-iron when the work was done. The plan of grading up one gangway over another is not adapted to dips of more than 20°.

There is one other plan which incurs the least expense and is the most likely to be adopted, namely: To drive an airway across

FIG. 119.



Plan No. 13.

the chutes between the gangway and the breast headings, as shown on plan No. 13 (Fig. 119).

This airway is not available when the breasts are being worked, because it is blocked with coal at each chute; but when breasts are full the ventilation can be conducted through the pillar headings. When the breasts are finished and the coal drawn out, if the pillars run, or falls of top choke the bottom of the breast and block the air-current, then the airway below the

heading is cleared at every chute, an air-tight stopping placed under it in the chute, and the chute lagged above it to keep it clear of anything falling down. By this method a return airway is maintained which the running of the pillars or falling of the top cannot affect. The airway can be driven as the breasts are emptied, but with timbered chutes it is best to drive it with the chutes, as blasting may damage the timber when breaking through into each chute.

Under all the conditions laid down in this question I would open the breasts as explained, and shown on plan No. 13 (Fig. 119).

Ques. 5.—In placing the "Battery" in a steep breast what precautions would you adopt to make it convenient for running out the coal and safe for the *starter?*

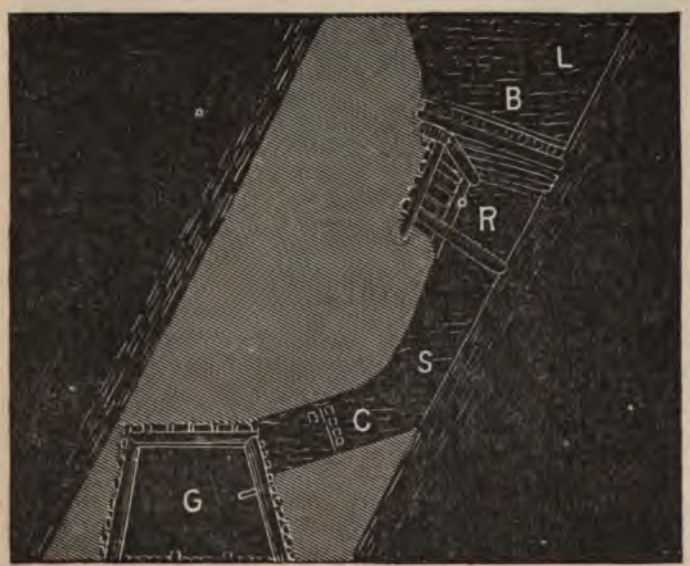
Ans.—The proper construction of a battery is very important, because upon its strength and stability the safety of the "starter" depends. When the "battery blocks" and dynamite placed upon a lump fails to break it, a hole has to be drilled and blasting resorted to, and the battery may have to stand some of the force of the blast. In starting the battery sometimes the whole mass of loose coal in the breast will come down several feet with a surge, and if the battery is not wedged tight and the timber tied by plank and spikes it may be "slewed," and fall into the chute with the starter under it. It is therefore imperative that the battery be of strong timber and properly stood.

In placing a battery under the conditions laid down in this question I would stand a set of double timber across the chute against the top slate, the collar as long as will reach across, and the legs spread two feet on the solid at each side to secure the foot-hold. One end of each battery prop is then placed in a foot-hole cut into the bottom rock. The other ends are laid on the collar, and wedged tight. Plank and slab are spiked across the props and the whole bound firmly together.

The top of the battery should be flat enough for fine coal to heap on it and form a cushion to save the battery when falls occur in the empty breast, as the battery may be used for skipping the pillars, or the timber may be saved.

A "booming-stick" should be placed under the battery and across the chute to prevent large pieces going down the chute, and a convenient plank fixed for the starter to stand on while breaking large pieces or working with the starting-bar.

FIG. 120.



Plan No. 14.

Plan 14 (Fig. 120) shows the position of the timber over the chute, which should be 8 to 12 inches diameter, according to the size of seam, the steepness of the dip, and the width of the chute.

G represents the gangway.
S " " chute.
B " " Battery.
L " " loose coal in breast.
R " " booming-stick.
C " " "check" for loading.

The "check" is a loose pole which works up and down behind two props and acts as a gate to regulate the rush into the car, and is pushed down to stop the coal when the car is full.

Ques. 6.—Make a sketch of the workings of a colliery opened by two shafts on a seam dipping less than 8°, showing the general plan of levels, headings, airways, and general arrangement of the ventilation, and positions of doors, overcasts, and air stoppings, with a brief description setting forth the general and important features of the same.

Ans.—Plan No. 15 (Fig. 121) represents such a colliery, *DC* is the downcast and main hoisting shaft, *UC* the upcast or air and auxiliary shaft with hoisting machinery as an outlet in case of accident. The arrows show the direction of the several currents: X marks the overcast where the returns cross over the intake currents for each split; *r* marks the position of regulators, placed in short air-courses to partially obstruct and reduce their current, and thus distribute the air in proportion to the requirements of each split; *WS* is the water lodgment or sump connected with the shaft by a small heading through which the suction pipes pass from the pump to the water.

By this plan of laying out the ground to be mined, each heading or panel is ventilated by a separate split or current of air. The method of working the coal from each district or panel may be varied to suit special conditions as to hardness of coal, top or bottom, as to dip or direction of the cleavage. The general plan would, however, remain the same.

It will be noticed that the air in each split may be distributed either upwards or downwards by placing a canvas air brattice across the entrance to the chambers, or across the heading road. There is no main door which, if open, cuts off the air from any part of the mine. The ventilation can be made ascensional in each tier of chambers, and gas will naturally ascend and be less likely to accumulate in the working faces. Where it enters the return it encounters the concentrated current to carry it down grade to the upcast shaft.

When the heading roads become long, new levels may be driven and the heading roads cut off to save rail, the coal being all hauled in the two headings next the shaft.

When a panel is worked, the chambers can be used as part of the return to keep them clear of standing gas, and also reduce the resistance of the air by lowering its velocity.

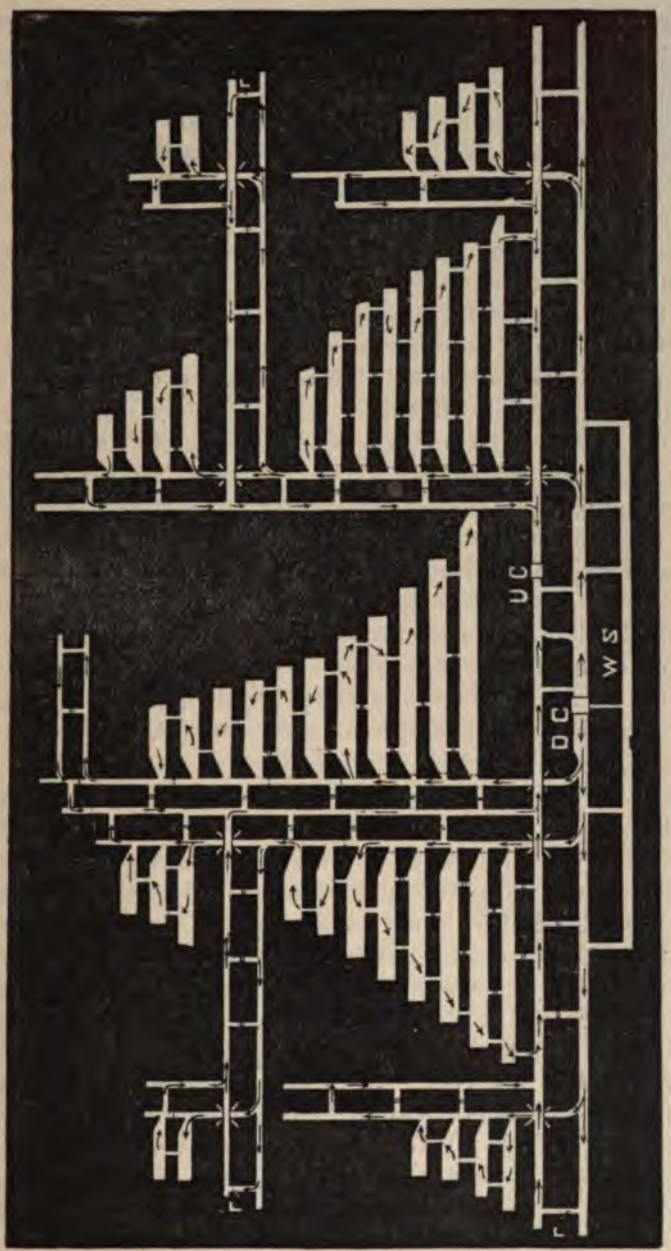


Fig. 121.

Plan No. 15.

When the boundary is reached the coal left to protect the roads and airways is mined backwards to the shafts; a large reservation pillar is left to protect the shafts from a general squeeze, as shown on the plan. The main and important feature of the plan is the separate split for each panel, without being dependent on doors, and the plan combines nearly all that is best in the science of mine ventilation.

Ques. 7.—Mining a seam of coal 16 feet thick, dipping 40°, with a very weak top, what plan would you adopt in working the breasts consistent with safety and economy?

Ans.—When all other plans failed to give satisfactory results, we would drive up chutes in pairs connected by headings for ventilation, secured with timber if necessary and slanted across the dip to make the grade of the chute about 30°, so that coal will just move without sheet-iron. When driven the required distance up, would mine the coal back in short sections, leaving the refuse and allowing the top to fall. The size and distance between the chutes would be determined by experiment.

Batteries of old props stood across the chute every few yards would hold the loose gob from following down the chute and keep the face clear. The expense per ton would be increased, but the tons per acre would also be increased, and if the interest of the landowner and operator were identical, both would be benefited.

Ques. 8.—What is meant by "Culm Flushing," and what are its objects?

Ans.—"Culm Flushing" is a system of mixing the breaker and bank culm with water and running it through pipes back into the old chambers in the mine. This culm packs tight and supports the roof during the removal of the pillars. It therefore enables a greater percentage of the coal to be won and lessens or prevents disturbance of the surface. Flushing has also been found useful as a means of stopping squeezes and fighting mine fires. In some cases it has been found cheaper to flush the breaker culm into the mine than to deposit it on the bank.

The culm is carried by a system of conveyors to a small hopper, usually an old oil barrel. Water is run into this hopper and the amount regulated according to the work to be

done. This mixture descends through a pipe, usually 4″ to 6″ diameter, into the mine; along the main gangways, up the chambers, through the cross-cuts, and into the chambers which are to be filled. At the foot or lower end of these chambers a dam or stopping is built which prevents the culm running out. If the pitch is light the pipe must be led close up to this dam. After the culm has filled up to the end of the pipe the flow is stopped, or conducted to another point, and a joint of pipe taken off, thus giving another section of the chamber to fill. Culm has been carried over a mile in this way.

Wrought-iron pipe is said to be the best for the purpose. With fresh water and small breaker culm, a pipe is said to last 18 months. When carrying larger bank culm mixed with ashes it last only about 6 months.

The size of the pipe, amount of head, and water required to flush a given quantity of culm are probably not subject to an accurate method of calculation. The following points however may lead to a better understanding of the subject. The presence of the culm in the water, no doubt, decreases the effective area of the pipe and increases its frictional resistance to the flow of water. We would therefore not expect a pipe to deliver as much water under a given head when running full of water and culm, as it would when running full of pure water. The average velocity of the culm in the pipe is probably not equal to the velocity of the water except where the descent is perpendicular or nearly so. If the pipe were horizontal the friction of the culm would keep it behind the water and, if it were going up hill, the effect of overcoming a part of the weight in addition to the friction would increase this "slip." If the culm had to be forced vertically upwards, the velocity of the water would theoretically have to be greater than the falling velocity of culm in water before it would have any effect at all. The velocity with which a lump of anthracite $\frac{1}{2}$ inch in diameter will fall in still water is somewhat less than 1 foot per second. This velocity is greater for larger particles of the same specific gravity and greater for particles of the same size but of greater specific gravity. As the "slip" probably remains nearly constant under different velocities, when other conditions are the same,

the number of pounds of water per pound of culm delivered would decrease when the velocity of the water increased.

The following is the performance of the culm plant at the Dodson colliery as given by Mr. Jas. B. Davis in "Mines and Minerals" (1898). The pipe going down the shaft and along the gangway is 6″ diameter. The branch pipes are 4″ and 5″ diameter. The length is 2500 feet. With this pipe it required 1½ lbs. of water to deliver 1 lb. of breaker culm, and 2 to 2½ lbs. of water to flush 1 lb. of bank culm. The culm in these cases probably did not have to be forced up hill. When the Dodson culm pipe was running west 1500 feet, up hill 300 feet until it was 90 feet above the shaft bottom, then east 700 feet to the discharge, it took 6 lbs. of water per pound of culm. The shaft is 400 feet deep but, as the amount of water was regulated to the work, we think the available head was probably much in excess of that required.

After the breasts are flushed they are allowed to remain until the water has drained off, then the work of removing the pillars is commenced. The pillars are removed by driving a road up the middle from the gangway. Usually it is necessary to leave a thin strip of coal next to the flushed culm to keep it from running out. Sometimes lagging must be resorted to for this purpose. After taking out the coal the roof settles for about one-quarter of the height of the vein. This breaks the timber, which must be replaced by shorter props. In some places at the Black Diamond Colliery, where the vein is 16 feet thick, it took 700 feet of timber for every 100 feet of breast pillar removed.

The finer materials are preferable to the larger for flushing, because they flush more easily, and pack better. In some localities, sand, gravel, ashes, and even loam, are flushed into old workings. Unmixed loam, however, is found to dry too slowly to be a desirable material for this purpose.

CHAPTER XI.

RAILROAD TRACKS, SWITCHES, FROGS, PULLEYS, ETC., IN AND AROUND A MINE.

Ques. 1.—State your views as to the importance of good roads in and around a mine, giving practical hints as to the best methods of laying down and maintaining the same.

Ans.—The tracks in and around a mine on which the coal is moved are next in importance to ventilation in the general equipment of a colliery. If the roads are bad, the coal cannot be moved cheaply or quickly; poor-laid tracks are a constant source of annoyance and delay; in fact, the success or failure financially of coal-mining operations is largely dependent on the methods adopted in laying, and the attention paid in maintaining, the roads.

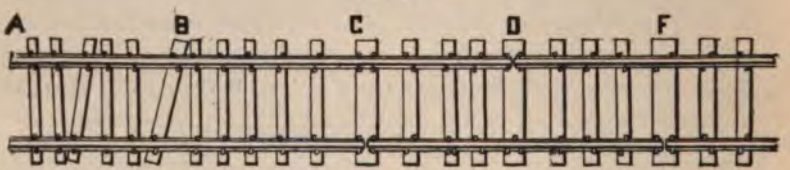
The first operation in track-laying proper is putting the cross-ties in position. Care should be taken to have them low enough, as it is much easier to raise than to lower them after the rail is down.

The broadest ties should be selected for the joints, and the joints broken as near halfway the length of the rail as practicable, thus placing the joint on one rail nearly opposite the middle of the other. For a main track, ties should not be over 30 inches from center to center, or, with ties having a face of 6 inches, 24 inches would be a proper distance apart. In rounding curves the broadest ends should be on the outside, and the ties placed radially to the curve. Sawn ties are preferable to hewn, because the face at both ends is in the same plane and fits to both rails, giving full bearing surface under each.

When the spike is driven, the tie should be held solid against the rail at both sides. The spikes should be driven home, but not hammered down too tight, as the heads are apt to crack off by bending the neck of the spike when the hook begins to

straighten on the bottom flange of the rail. Another important thing is locating the spikes. If they are driven in the middle of the tie, it is apt to rock; and the spikes being in the same layer of the timber, the ties are apt to split. Therefore, the spikes should be placed about 1 inch from the edge of the face on opposite sides, thus holding them solid against the bottom of the rail. Another thing to be observed in spiking is to have the outside and the inside spikes of both rails on the same side of the tie. This is to prevent the tie from slipping along the rail and getting slant across the track, as shown in Fig. 122. In the diagram, from *A* to *B* the spikes are wrong; from *B* to *C* they are right. The difference between right and wrong can be understood at a glance.

FIG. 122.



Joint ties should be broad, so as to give ample space for spiking both rails on the one tie. The end of the tie opposite the joint should have four, or at least three, spikes, as shown on the diagram at *C* and *D*. This is to hold it up at both sides and prevent twisting or slipping along the rail, and causing a notch on the joint. It should never be left with only two spikes, as shown at *F*.

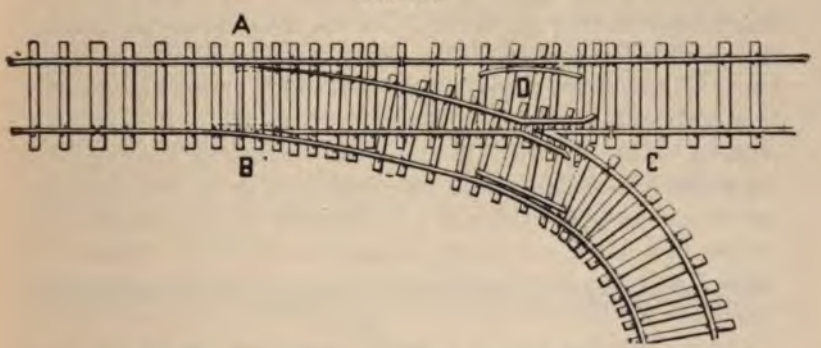
Ques. 2.—Describe in detail how you would proceed with the work in laying down a switch or branch from a mine track.

Ans.—When the branch road has been shaped off, a frog with the proper angle will be necessary. Generally the place can be opened to suit the frog, and driven on a curve to bring it round to the proper bearing. It is best to place the frog on the track and slide it along the rail until its angle points in the proper direction, then mark the rail so that the ties on which the frog is to be spiked can be put in before tearing up the road and stopping the traffic. By proper management, switches can often be put in without stopping the main road more than one hour.

Next measure the length from the frog to the points of the switch. If the road you are branching from is straight, one switch rail will be a few inches longer than the other. Branching from the inside of a curve requires longer rails, and their difference is more; while branching off the outside of a curve requires a shorter switch, and the rails may be equal or nearly so.

In measuring the length of the cross-rail, hold the tape at the frog, stretch it along to where the point of the cross-rail fits against the track, stick in a few pegs to keep the tape to the proper curve, so that it points straight through the frog at one end, and curves around so as to lead on to the track-rail at the other end, with as little angle as practicable. This can be done near enough by the eye after a little practice. Where small cars

FIG. 123.



are used, and the switch is a branch into a chamber, the switch-rails are tapered at the end, which forms the points, and are spiked so as to leave a small opening for the flange to pass through, as shown at *A* and *B*, Fig. 123.

A plate of iron nailed under the point so as to bear the flange carries the wheel over the space. This works very well with small cars where the loaded trip is always going one way; as the empty trip going in, facing the points, is not apt to run into the switches, but has to be "slewed" in. The other ends of the switch-rails are bent so as to form the wings of the frog, while the point of the frog-crossing is formed of two pieces of rail welded in a V-shape as shown at *C* and *D*, Fig. 123. This is perhaps

the simplest, but not the best kind of switch for large cars or heavy traffic. For such switches common short ties are preferable to long ones, which hold both tracks, as used for permanent branches; because they can be removed when the chamber is finished, and the main road left as before. It is good practice to have the frog raised from 1 to 1½ inches above the level of the opposite rails, and from ¾ to 1 inch wider than the gauge at the frog. Mine cars have the wheels loose on the axle, and often with considerable play. Therefore a guide-rail opposite the point of the frog is less efficient on mine tracks than on railroads. Having the track wide and the frog high tends to swing the car against the rail and away from the point of the frog. This raising of the frog is easiest done with short ties placed alternately in each track, as shown in Fig. 123.

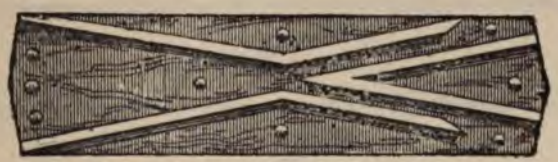
With long ties the rails have to be sunk in notches cut into the timber; but for permanent jobs the long ties are preferable, as they bind the whole switch together. The ties once in position, spiking down the rails is very simple. Rails for switches and curves should be bent so as to lie in shape before being spiked. The practice of bending with sprags and holding rails in shape with spikes is not for permanent work; the vibration of the cars and the spring of the rail, work against the spike till it becomes loose, or cuts off the spike at the hook. When the switch is down, the ties should be packed and tamped tight, especially under the frog and points. Another important thing in laying switches is to have the points higher than the rail they lead on to. Many of the old wheels become grooved, and if the point is low it will not carry the tread of the wheel over on to the rail, and the wheel is apt to jump the track or push the rail out of gauge, especially with heavy cars. The points can be raised by notching the main rails into the ties.

Ques. 3.—Describe in detail the different kinds of frogs, latches, and other attachments belonging to switches with which you are acquainted, the conditions to which each is adapted, and the cheapest and quickest means of supplying the same at the mines, illustrating your ideas by diagrams?

Ans.—In arranging the roads in a mine the main features to be observed are simplicity and strength, so that they can be

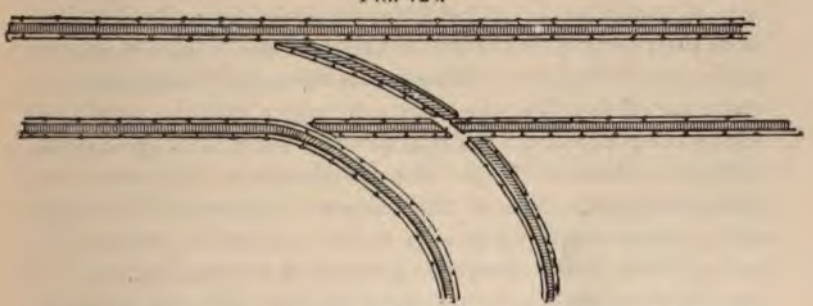
easily operated and need little attention or repairs. Frogs, as shown in Fig. 123, are made in the shop at the mines. The blacksmith takes a piece of rail from 5 to 8 feet long. At a point a foot from the middle a piece of the bottom flange is cut out on one side. When this part of the rail is red-hot it is bent sharply in an acute angle, and the point is then hammered into a V-shape, as shown in the figure. The object of the cutting

FIG. 124.



out of the flange away from the middle is to make the one leg longer than the other, so as to break joint on the ties. This kind of frog does very well for the small cars used in thin, flat seams, but not for a permanent track. A better class of frog is made of a solid cast-iron plate, with a track cast on its surface, as shown in Fig. 124. This frog is fastened down by rag-spikes driven through the bed-plate in counter-sunk holes.

FIG. 125.



Another temporary type of frog is made by merely butting the switch cross-rail up against the main-rail, and cutting out a slit in the top of each to allow the flange of the wheel to pass through, as shown in Fig. 125.

The cross-latch is another substitute for a frog. It is merely a

short piece of rail, or bar-iron, jointed on a strong pivot-bolt at one end, which carries the switch cross-rail over the rail of the main track, and lies folded back when not in use. It has to be moved into position by hand. When used it lies back against the wing of the switch-rail to support it in place while the car passes over. Fig. 126 represents it in place for the branch, and the dotted lines, *AB*, indicate its position when not in use.

FIG. 126.

The weak points in this device are its liability to damage from neglect. It is only used for branching from a level up grade. By its crossing over the main track it elevates the outside of the curve on the branch road, and allows a gradual rise from the points of the switch. Cross-latches require to be very secure and well fitted. Where they are used, the cars are generally run down with the wheels spragged, and the cross-latch resists the strain of the car rounding the curve, at a good speed, with the wheels sliding. The switch-rail is raised on liner pieces securely spiked on the ties, and requires to be braced along the outside by planking nailed on the ties of the main road. With heavy cars, on a grade of 6 or 7 degrees, for long use and much traffic the cross-latch does not pay. It is cheaper to blast up the bottom rock for a few feet up the grade and put down a frog-switch, bending the rail to fit the change of grade on the branch road.

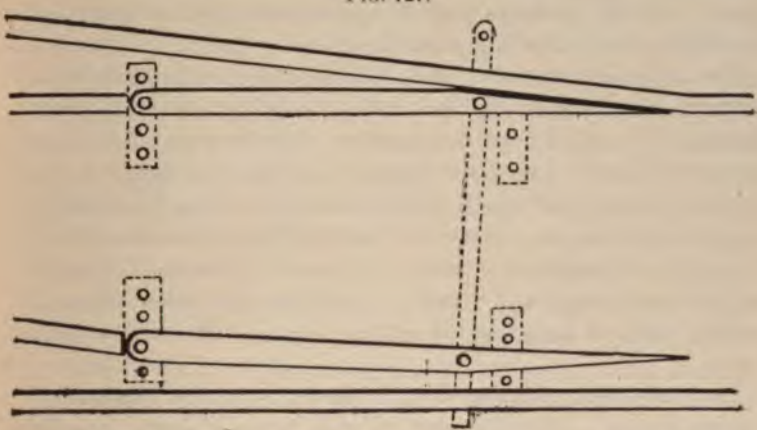
The best kind of frogs for mines is made of rails riveted to a bed-plate of boiler-iron, similar to the permanent frog used on

railroads, the end of the rails being punched for strap-joints, so that the whole switch can be bound firmly together.

Ques. 4.—Describe switch-points, how they should be fitted, and what forms you prefer.

Ans.—Where light cars are used we prefer fixed points, as shown in Fig. 123; where the cars are large, the weight heavy, and the traffic considerable, we prefer tongues made of bar-iron as shown in Fig. 127.

FIG. 127.



Tongues for switches are from 3 to 4 feet long, made of bar-iron, from $1\frac{1}{2}$ to 2 inches square. When the switch is laid, a thin lath cut to a pattern enables the blacksmith to shape the tongues so as to be a good fit. Tongues operated by a handle or springs have studs inserted at the taper, the shifting-rod passes under the rail, and is punched for the studs; a spring catter-pin or nut secures the rod and prevents it from dropping off the stud. The heel-bolt is secured by plates nailed on the ties, and the taper end moves on bearing-plates, fastened in like manner, as shown in Fig. 127. On slopes, double track sidings, and other important places we prefer a split-switch. Split-switches, at places where the cars are to pass always in the same direction, should be made of steel-rail, so as to act as springs. When the flange of the wheel forces its way through by pushing back the point, the rail, being of steel and spiked to within

3 or 4 feet of the point, springs back and closes the switch without any attachment further than bearing-plates on which it slides. In case a car should have to be backed through, a block or splinter inserted behind the point will hold it open for that purpose. These split-switch rails have to be dressed in the shop. The taper, which may be from 15 to 24 inches long, is laid off by a scribe line on both sides. This is to preserve the center-web of the rail to the extremity of the switch-point. Both sides of the ball or top of the rail are sliced off with a hot-cutter, and the bottom flange on the side which fits up against the rail is sliced off in the same manner.

The taper on both sides has to be laid over so as to fit the lath patterns, for the right and left sides of the switch in the same manner as in fitting bar-iron tongues. If the switch has to be operated by hand, lugs are riveted through the web and over the bottom flange on the inside. The switch bars are bolted to these lugs, and the bar connected to a short tumbler handle, similar to the monkey levers used in yard switches on railroads. A vertical lever standing at the side of the track is dangerous for a passing team of mules in the confined spaces allowed on mine tracks.

The stub switch is often used at mines, especially around the outside works. It is specially adapted for some places and is very simple; but as it cuts both rails with an open joint there is always a jar in passing over, which is very severe on the running gear under heavy cars. For general purposes, we prefer a well-laid and neatly fitted split-switch.

Ques. 5.—Explain how you would arrange the tracks and switches at the different lifts on a slope, the coal being 4 feet thick with a weak top, the dip over 20°, and the load on the rope 5 cars each trip.

Ans.—To hoist coal cheaply and quickly requires the tracks and switches arranged so that a stout boy can send up 1000 tons in 8 hours. The empty trip must run well on to the side track to give plenty of slack. The *chainer*, by a quick movement of a few feet, can change the rope from the empty to the loaded trip, and give the signal in less time than it takes to reverse the engines.

To do this properly, a working draft of the track should be made, and the space for the switch and curves taken out by measurement. Breaking off from the side of a slope by guess or only guided by the eye, is apt to result in useless space being left at the sides, which weakens the pillars and top, causes extra timbering, makes an unworkmanlike job, and remains a standing eyesore until the lift is finished and the rails are withdrawn.

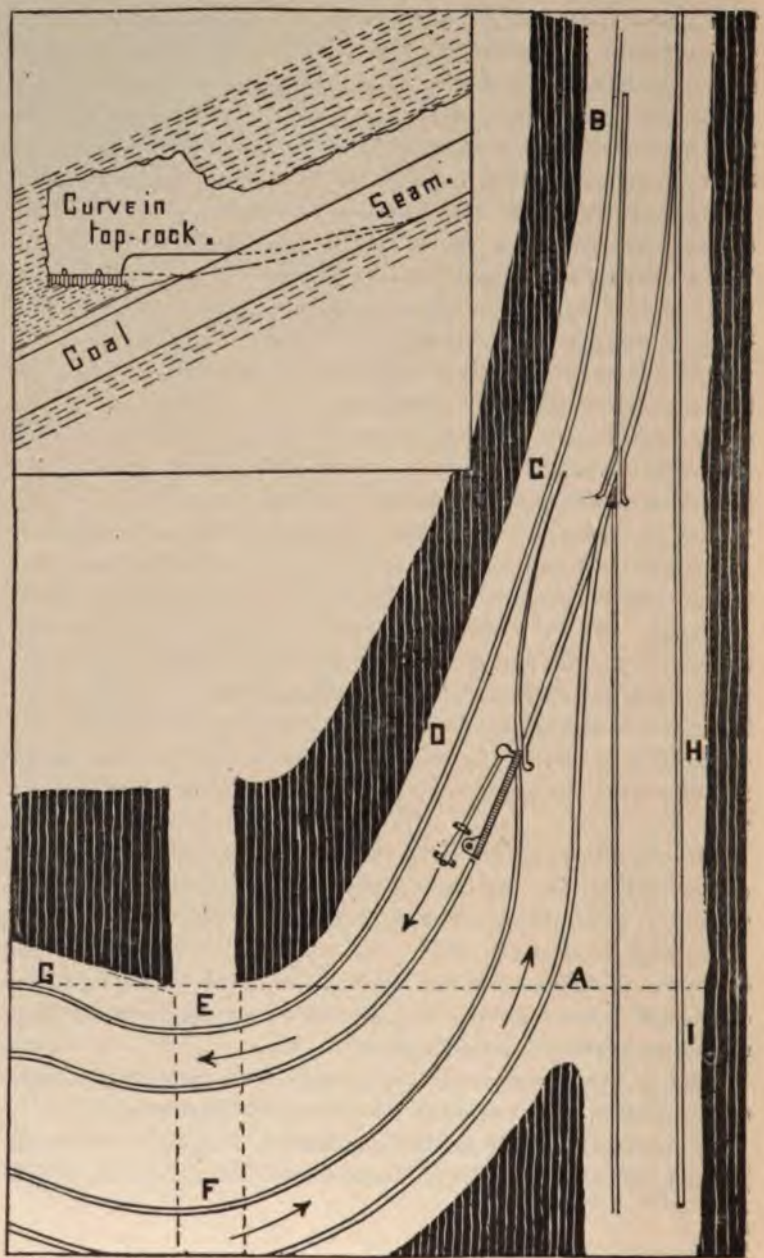
Fig. 128 represents the plan of the switches, with a profile, showing the space for the curves taken out above the seam. The reason for this arrangement is because the empty track has to be graded to run the empty trip in off the slope, and the loaded track graded to run the loaded trip out to the side of the slope. Therefore, if the curves were driven in the coal, the lowest point or change of grade would be round at the inside end of the curve, and away from the slope. The empty cars might run in and drag the rope with them, but the loaded trip would be standing nearly at right angles to the pull of the rope, while the rope would be pulling round the corner of the pillar. A snatch-block might hold the rope in line to start the cars, but it has to be on the track, and the rope chain has to be put on it by hand. We have often seen it used, but never saw a landing of this kind work satisfactorily or fit for handling much coal. By curving round horizontally over the coal seam into the overlying strata, the lift or change of grade on the loaded track can be brought round nearly to the slope, and the front car faced round so that the rope in starting the load has nearly a straight pull.

The first branch, *BC* (Fig. 128), may be either a stub or split-switch with a tumbler handle on the opposite side of the track. It grades with the slope down to the frog *C*, its direction being slightly across the dip. The side *BCD* is higher than the frog when levelled across the track; therefore in order to get the inside rail lower than the frog, the bottom slate along the side should be blasted out the depth of the ties.

This is to counteract the centrifugal force tending to throw the rapidly moving cars over the outside of the curve.

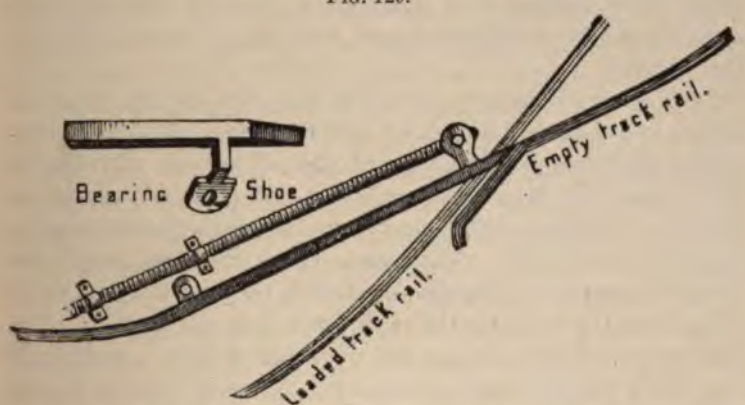
From the frog *C* the grade diminishes to *D*. Between *D* and *E* the rails are bent in the profile or what is termed *dished*, which

FIG. 128.



reduces the grade to a uniform dip inwards of 2 or 3 degrees, just sufficient to prevent the front car from stopping suddenly and causing those behind to bump and jump the track. The grade on the loaded track diminishes as it goes down to the lowest point or middle of the dish at *A*. At *A* the grade changes to an up grade to *F* round the curve and back along the double track, the opposite grade of the empty track meeting it nearly half way back the siding. The grade outwards on the loaded track should be such that when a trip is lifted out and up the slope, the next would move into the same position for the next hoist, the front car standing about the position of the arrow at *A*. When the rope tightens the first car mounts on to the grade, the others follow in, with the couplings slack enough to allow the car to swing freely round, and only add their weight to the trip as they mount the grade.

FIG. 129.



Instead of a frog at *D* the empty track rail is carried over the other about 5 inches by a spring cross latch 5 or 6 feet long, as shown in detail at Fig. 129. This allows of the empty track commencing to lose grade at the point *C*, and gives a greater difference of height between the two tracks when they reach the lift at *A*, which is all the elevation available to allow the grading on both tracks. A frog at *D* would keep both tracks in the same plane to that point, and make the distance to the *dish* too

short to give difference of elevation sufficient to grade both tracks in opposite directions. The empty curve is carried on a cribbing within 8 inches of the rail, leaving the space between the roads level with the loaded track for the chainer to move about the cars while at work.

Where the rock will allow of a very wide opening being made, all the switches can be placed along the side of the slope, with frog crossings, but it takes too wide a space for these switches to stand secure with a weak roof. Large expensive timber would secure it, but a runaway trip may wreck the timber. We prefer the cross-latch, or the end of the rail shaped to fit as such, with a shoe riveted to it to hold it up, and 6 feet of it left unspiked on the inside so that the tread of the wheel will push it back to get through. It must be a steel rail to spring back again and work well, and is simpler than the cross-latch and push-spring shown in Fig. 129. We have used both successfully. The bearing shoe slides on a plate of iron, which is spiked along the outside and level with the top of the rail of the loaded track.

It is generally best to have the rails bent in the profile or *dished* before being taken down into the mine. The side bending for the curve can be done with the rail-bender. If the dish is not a good fit, it can be changed by laying the rail across two blocks and striking with a heavy hammer while under a bind or pressure from a lever. The switch rail of the empty track from *C* to *D* is raised on pieces rag-spiked to the ties, where it turns off to form the wing for the cross-latch to spring against. It is continued alongside the loaded track rail to form a guide, and the end is turned down into the tie, to prevent the slack rope from fouling on it.

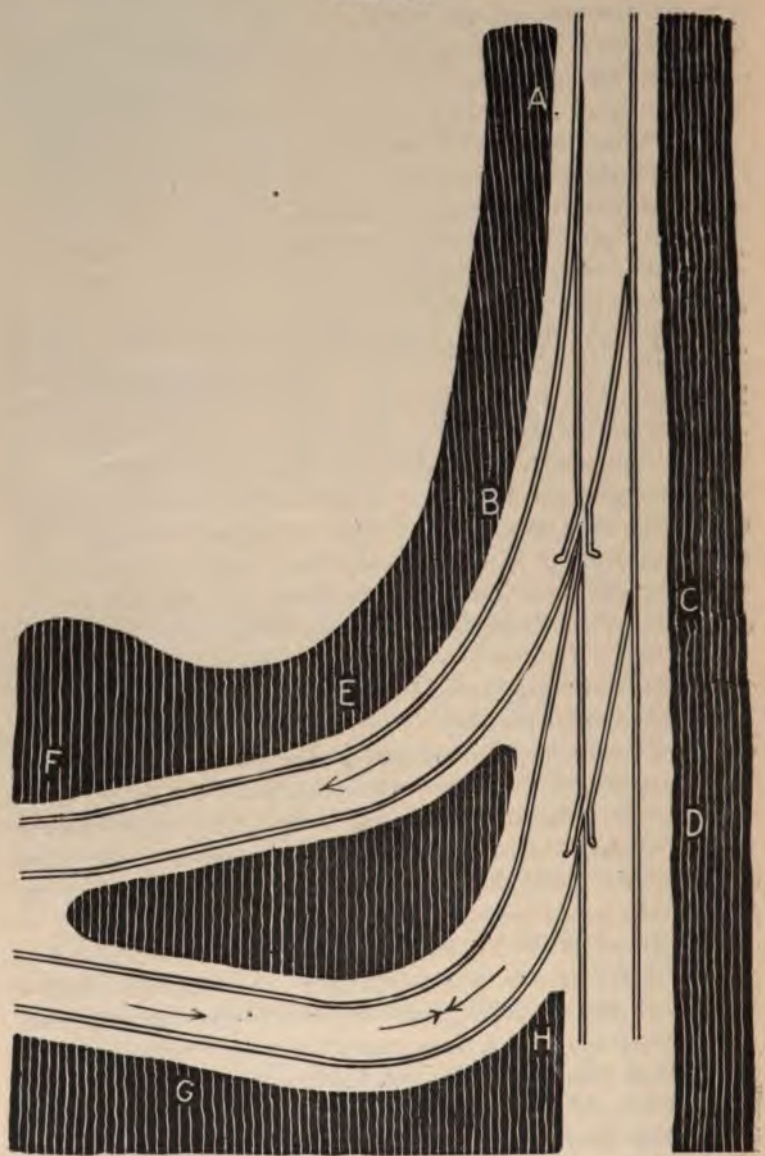
We advise those interested to pay some attention to this subject, as they may find some unexpected difficulty in their first job of putting down the switches and curves on a slope dipping over 20° with a bad top.

Ques. 6.—Explain how you would arrange the tracks and switches for a lift on a slope dipping 10° or less, the seam of the coal thin, the strata over it very weak, and the load to be hoisted 6 cars each trip.

Ans.—The conditions as stated are similar to Question 5, except in the dip, but the difference of dip means much. The difficulty on a steep slope is to change the grade in such a short distance, and get sufficient difference of height between the tracks to allow them being graded in both directions. The answer to Question 5, is an attempt to explain these difficulties, and the methods to overcome them, hence it is much lengthier than the apparent importance of so simple a looking thing would warrant. On the flat slope with the same plan of tracks, the empty trip would not drag the rope far enough to reach the loaded one, the same length of rope at the same time being much worse to drag. We have seen slopes where there were 20 feet of chain attached to the rope. The empty cars going down were hitched on close to the rope, and the chain thrown into the car. At the bottom the 20 feet of chain gave slack to reach the load. This plan, however, does not remove the difficulty of the rope having to pull round the corner, and lift cars standing several yards away from, and at right angles to, the slope. We found this kind of a lift, which had been made on a slope, gave us much annoyance, and was both slow and expensive to work. We therefore arranged the next lift according to the plan shown in Fig. 130. To open a new lift under the stated conditions we would open the single-track curve *DHG*, with just space on the upper side to pass the cars. The rails are slightly dished at *H*. The first loaded car should stand in the position of the arrow, and be out on the side of the slope just clear of the frog. From *H* the grade is up past *G* into the loaded siding. The empty track is made by driving a narrow curve up grade through the corner of the pillar, coming out 8 or 10 yards up the slope. From *F* the tracks are together.

On a dip of 10°, if the distance between the frogs *B* and *D* is 24 feet, it will give a vertical difference in height of nearly 4 feet, which will be ample grade on both tracks. By this plan the rope will be dragged far enough to be carried round the front of the small triangular pillar and hitched to the load. The car standing out and well round at *H*, makes the rope have a direct pull. This plan works satisfactorily. The distance the rope has to be changed is one disadvantage, but on a double-track

FIG. 130.



slope this can be done as quickly as the change on top. The two switches from the slope are not desirable. All things considered, this plan will compare very favorably with any we have seen for the handling of coal on a flat slope with a weak top. The openings being narrow are easily secured, the cars move into position by gravity, one person can manage the lift and send up a large quantity of coal. With a very thin seam a little of the top over the lower curve may have to be taken down to improve the curve, but this plan when properly carried out is the best and cheapest to fulfil the conditions of this question.

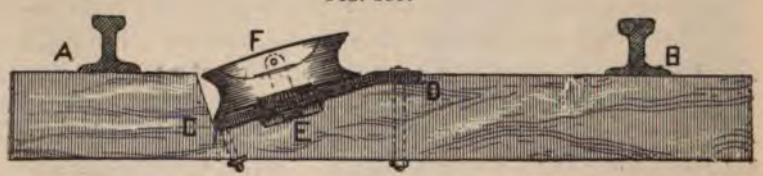
Ques. 7.—Explain your method of keeping the rope in line. Show by a diagram the best device for a side pulley to hold the rope over in the centre of the track.

Ans.—This question bears upon another difficulty which a mine foreman may encounter. It looks very simple, except to those who have had to deal with it practically. We have used blocks of hard wood, car wheels with the axle stub inserted in the ties, cone-shaped wooden pulleys with a sheet-iron flange, and cone-shaped side pulleys which were supposed to be made expressly for the purpose. None of these devices worked satisfactorily, they being only makeshifts. When the rope got accidentally on the wrong side it could not get out. The rope was stripped until the car came up close, when it was either torn out or the rope damaged. Most of these devices required the chainer to go up the slope to the upper switch points and adjust the rope before signaling, or the rope was in danger of being badly damaged for 50 feet at the end. The main difficulty was to get a side pulley high enough to catch the rope as the trip came down, and still be low enough for the cars passing over. The wheel required to be a tight fit so that it would not lift up and allow the rope to get under it, and be shaped so that the rope would wrench itself out over it when on the wrong side. None of the devices fulfilled these conditions, and all were a constant source of annoyance and expense. The least inattention on the part of the chainer might result in ripping a strand out of a new rope for 50 or 60 feet, stopping the work until the damaged piece was cut off, and the rope resocketed. The expense and delay in such cases are a very serious matter where

large quantities of coal have to be hoisted. This is what makes the subject of side pulleys a question of much importance.

Fig. 131 is a sketch of a side pulley, which we got made, and found better adapted to the purpose than any devices we have tried. *C D* is a cast-iron bed-plate ¾-inch thick, bolted into a notch cut in the tie. A boss is cast on both sides at *E* to give surface hold in the plate for the axle-pin *F*. The upper boss is faced for a bearing, and the pulley has center bosses faced to fit. The pin is either shrunk in, or the plate is cast around it. The pin should be mild steel, to bend rather than snap off. It should fit accurately, and not be less than 1½ inches in diameter. A washer and catter-pin hold it down. The object of the sunk boss of the pulley is to get the top of the pin down level with the rim, and the back is sunk deep enough into the tie to cover

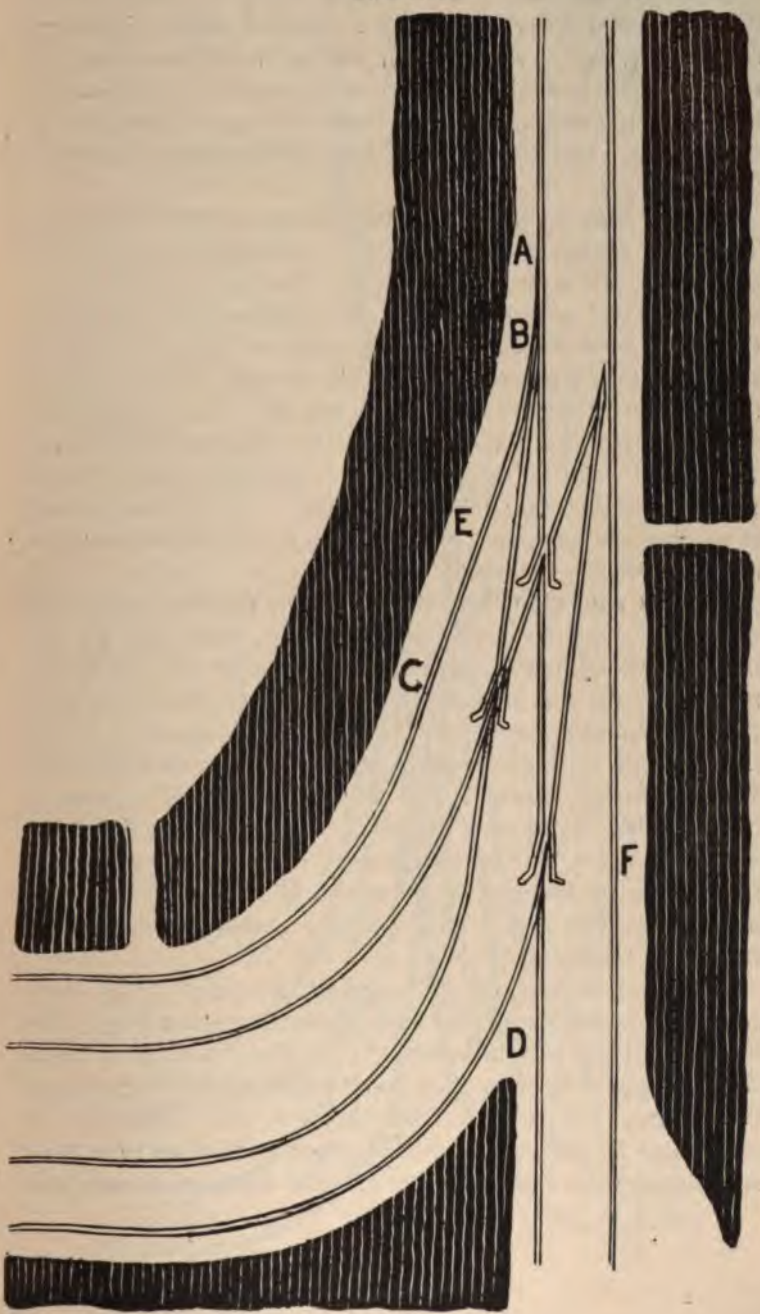
Fig. 131.



it. Therefore, if the rope should miss it, and be led off to the side by the cars leaving the slope line, when it tightens it will slip back over the pulley and right itself without tearing out wires or doing much harm to the rope. If it is too low to catch when the empty trip passes down, it can be raised by notching the rails down in the tie. The first pulley is placed above the switches. The proper place is found by watching where the rope begins to follow off with the cars. When the rope drops into the first, another should be placed to catch the rope where it turns off; generally two will be sufficient. These pulleys should be cleaned out and oiled every day, as upon their being kept in proper working condition depends to a great extent the life of the rope. Side pulleys are a thing of more importance than the attention which is generally paid to them would indicate.

Ques. 8.—If you have any other special points or plans on this subject, explain the same by diagrams.

FIG. 132.



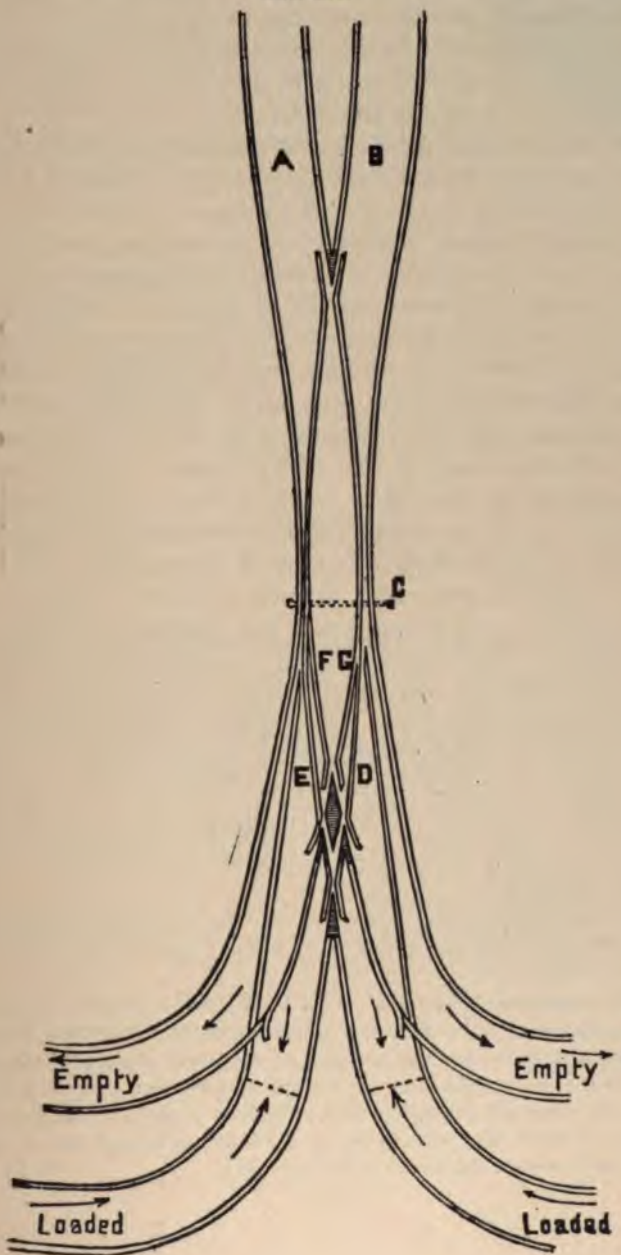
Ans.—We have seen many plans of tracks at the top and at the bottom and different levels of slopes and shafts. Those we have endeavored to show are all with a view of requiring the smallest space possible to do the work properly. It is easy to have finely-planned tracks where there is plenty of room, but in a mine with a bad top the great thing to be observed is to plan for little space.

Another plan for branching from a slope is shown in Fig. 132. The upper branch is a throw-switch. The other, being for the load track, is a split spring-switch. This plan is very simple and can be laid in a small space. The distance from the frog at *C* to the dish or lift on the loaded track at *D* is too short to admit of much difference in elevation between the tracks, hence there is more difficulty in handling the cars. The double frogs on the slope track are another objection. Still, on dips between 10° and 20°, and for small trips of 3 cars, this plan will work much better than many that are adopted. It has the advantage of needing less space than others, which is very important where the top is weak.

Fig. 133 shows another plan which we adopted and found very satisfactory for a double-track slope, where all the coal from several seams was hoisted from one level, they being connected by horizontal cross-cut tunnels. *AB* are the slope tracks, *C* is a double stub-switch, with 3 moves, so as to connect both tracks. This is to allow the ropes to work from either side. With the switch handle in the middle notch, each rope works its own side. By moving the switch the cars can be hoisted or lowered into any side by each rope. The switch is moved by the chainer at the bottom by means of a connecting-rod and ball-crank. The empty track-rail is a spring cross-latch over the loaded track-rail, the same as in Fig. 128.

The other switches at *FG* and *ED* are split-spring steel-switches, and stand right for both ways, being self-acting. The frogs are riveted on a bed-plate. This plan for a dip of 12° to 20°, for large 3-ton cars, gave good results where a large quantity of coal was hoisted from the one main landing. Its advantages for such work are very apparent to those practically acquainted with such matters. It was designed to condense

FIG. 133.



the tracks to suit a bad top and very expensive timber, which limited the space, and the tracks had to be planned accordingly.

Ques. 9.—Explain by a diagram how you would arrange the tracks and switches at the top of a slope for handling large quantities of coal quickly and cheaply.

Ans. Slopes being driven in the seam of coal from the outcrop down may be located so that there is not sufficient space for a long landing in the line of the slope. Therefore the top arrangement is largely dependent on the topographical features of the ground, the location of streams, and railroads, etc.

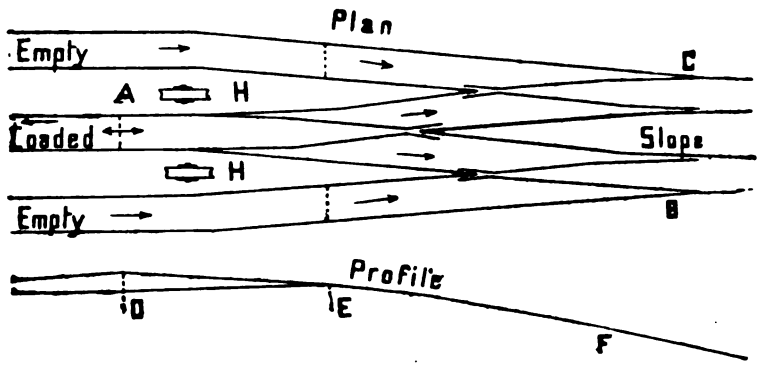
All tops should have a straight landing twice as long as the trip to be handled. For a single-track slope, where it can be arranged to back-switch the trip past the slope into the loaded siding, and hoist the empty trip out of the empty track on to the slope track, and back-switch it down the slope; this plan is the quickest and cheapest that we have tried. The cars require no hand-pushing, the loaded track-switch goes off highest up, which gives grade to run back, while the empty cars are brought in at a lower level, much like what is shown in Fig. 130. In fact it is just the same plan, only the loads are hoisted up and let run back on the switch, while the operation is reversed for the empties.* The engine does all the work, except changing the rope. This plan can be applied to a double-track slope, if the hoisting machinery is of the type used at many places in the bituminous coal fields. There is a separate drum for each rope, which is worked by means of clutches or friction gearing. This allows the cars to be run back without giving slack on the other rope, but it is not so simple and handy as with a single-track slope.

Fig. 134 shows a well-arranged top for a double-track slope. *B C* are spring split-switches, which lead into the loaded track

* With some styles of cars used in the bituminous regions this cannot conveniently be done. With such cars the end at which the car is intended to be loaded is made low for that purpose and the gate is placed at the opposite end. As the loading end faces the working face in the rooms, the gate end necessarily comes out of the slope first or in front of the car. As this is the direction in which they must be fed to the tipple no arrangement of tracks which would reverse the car can be accomplished. F. E. B.

from both slope tracks. The empty tracks are on the main deck. The loaded track is on a raised bench. The grade from the slope over the bend is continued up until it is three feet above the others. This is to allow grade from *A* to the tipple. The empty track is graded down towards the bend, just sufficient to cause the empty cars to move. They are run down against a stop-block, where the trip is made up and coupled.

FIG. 134.



The rope being attached, the block is withdrawn by means of a lever, when the trip moves off, tightens the rope, and runs over the bend down the slope. *H H* shows about the position of the first bearing sheave, when the empty trip goes over the bend. The rope should fall into the groove of the sheave automatically. This is very important, and the exact position is found by experiment. If the slope is steep, the grade must be reduced on the trestle-work below the bend for a distance equal to the length of the trip. This is to prevent the front car from being dragged off the track as it mounts over the bend, by the rope pulling a little off the straight line, and the weight of the cars behind, acting as the other pull. For a double-track slope to handle large quantities of coal we would adopt this plan in preference to any we are acquainted with, other things being equal.

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